ENCE LIBRARY 182 ✓

DEC 10 1932

Engineering and Mining Journal

WITH WHICH IS CONSOLIDATED ENGINEERING & MINING WORLD

NEW YORK LONDON

Gold-copper smelting plant at Rönnskär, Sweden

December, 1932

A McGraw-Hill Publication

Price 50 Cents, or 2s. 6d. Per Copy

THE DORR FX CLASSIFIER SETS PERFORMANCE RECORD

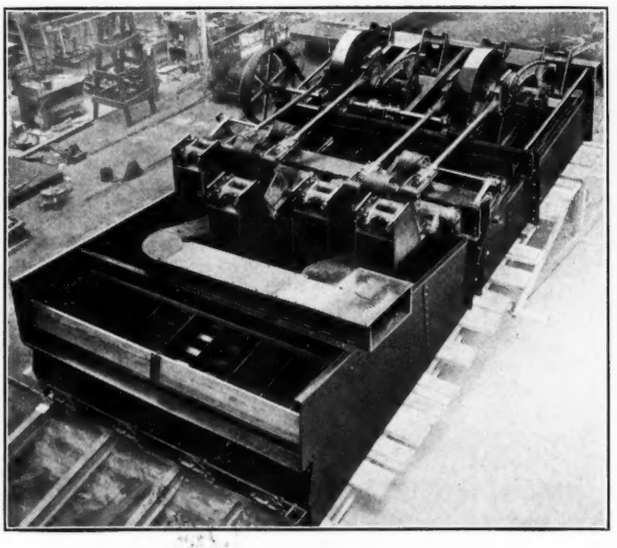
3 Times Former Sand Raking Capacity
2 Times the Former Machine Weight
30 to 40% More Grinding Mill Capacity

"The Red Devil" was what they called the first Dorr FX Classifier at Nevada Con—for she was installed in a rush with her priming coat of red lead unrelieved by a final coat of black. "The Pride of the Andes" was aptly applied to the second one at Braden—because she was the huskiest piece of classification equipment ever carried up the long steep grade from the coast, 8000 feet below.

Before these units were placed on a production basis, the design had to prove itself under every conceivable condition of load and speed. It took two years for the FX to graduate from the "university of hard knocks" and the curriculum included concentrated shop tests, semi-works demonstration and finally, full commercial operation.

The Braden unit combines 47,000 lbs. of double extra heavy construction with a 12 ft. width of rake—twice the weight of any classifier of this size ever built before. The extra deep blades easily handle 12,000 tons of sand per day—1,000 tons per foot of rake width—three times as much as ever handled before. As a result the capacity of the 8′ x 12′ ball mill in circuit with it was increased 30 to 40 per cent.

Nevada Con's "Red Devil" and Braden's "Pride of the Andes" are the forerunners of a long line of Dorr FX Classifiers, now offered in several sizes for heavy duty service. They will run as fast as 50 strokes per minute without vibration. They will handle three times as much sand as their predecessors. **They will increase the capacity of your circuit or the fineness of your grind, without demanding more space for classification.**



The Dorr FX Classifier that set this performance record.

Why not install a Dorr FX **now** on your test section? You'll be surprised how it will change your ideas of classification and point the way to complete modernization later.

DORRCO

THE DORR COMPANY, INC.

ENGINEERS • 247 PARK AVENUE, NEW YORK

DENVER CHICAGO LOS ANGELES TORONTO

Affiliated with OLIVER UNITED FILTERS INC. Engineers

Dorr technical services and equipment are available from the following companies:

HOLLAND: Dorr-Oliver N. V. The Hague
FRANCE: Soc. Dorr-Oliver, Paris
ENGLAND: Dorr-Oliver Company Ltd., London
AUSTRALIA: Crossle & Duff Pty. Ltd., Melbourne
SOUTH AFRICA: Edward L. Bateman Pty. Ltd., Johannesburg
GERMANY: Dorr-Oliver Gesellschaft, m. b. H. Berlin
JAPAN: Andrews & George Co. Inc., Tokyo

Engineering and Mining Journal, December, 1932. Vol. 133, No. 12. Published monthly by the McGraw-Hill Publishing Company, Inc., 330 West 42d St., New York City, N. Y., U. S. A. Subscription rates, United States, $3 per annum; Canada and other countries in the Americas outside the United States, $4 per annum; elsewhere $5 or 25 shillings per annum. Price per copy, 50 cents or 2s. 6d. Entered as second-class matter June 20, 1879, at the Post Office, New York, N. Y., under the act of March 3, 1879. Printed in U. S. A.

Engineering and Mining Journal

with which is consolidated

Engineering & Mining World

December, 1932 — Vol. 133—No. 12

Inventive Opportunity And First Principles

THE DISCLOSURE OF FACT by research has proceeded rapidly in recent years, but rarely does the would-be inventor realize the vastness of the field that awaits exploitation by a study of fundamental principles and elementary reactions. Transport of granular material is effected, for instance, largely by its carriage, with or without the aid of a fluid medium; yet a simple principle seems to have passed unnoticed until recently—that an arching effect could be produced in such material that would permit its elevation or horizontal movement with minimum expenditure of power. The system, known as the Redler, was described in the November issue. Flights of metal projections, occupying a negligible proportion of the cross-sectional area of the conveyor trough or pipe, and connected by endless cable, impart to the material a movement *en masse*, the effect being an absence of positional change of one particle with respect to its neighbor that seems almost unbelievable to those who have not observed the system in operation. The Redler method of conveying, used successfully in flour mills, is now being adopted for transport of crushed ore; and engineers and millmen will watch further developments and applications with keen interest.

Several months ago this publication commented favorably on a proposal to effect rock crushing by gravity fall, controlled and operated under sound engineering principles. Although the scope of so simple and apparently effective a system has not yet been fully demonstrated, although the applicability of the principle in industrial operations remains to be proved with all the various types of raw material that comprise the minerals of commerce, sufficient work has been done and sufficient results have been obtained to demonstrate that important invention can result from an inquisitiveness as to the practical application of first principles. The Hadsel mill is operating successfully and economically at the Beebe property, in California, as an article in this issue, by Mr. John B. Huttl, demonstrates. Its success is tribute to the ingenuity of its inventor and to the acumen of its sponsors.

Crushing and conveying comprise two important operations in ore treatment. *Engineering and Mining Journal* is privileged to bring to the attention of its readers the details of an invention in each of these phases. This augurs well for the future technical progress of the industry. Not the least significant feature of these advances is that both come from engineers whose names are not to be found in mining and metallurgical directories; so they have probably tackled the problems involved without the unconscious bias toward proved practice that retards the adventurous spirit of so many specialists. Nor are these inventors connected with any agency that sponsors research and is well equipped, with scientists and apparatus, to make important contributions to the art. Each has provided practical help and inspiration that will prompt the stimulation of interest among others of similar intellectual restlessness, irrespective of vocation or resources.

Economic Recovery and The Capital Goods Industries

ALTHOUGH commodity prices have fluctuated since their downward trend was reversed, optimism has replaced pessimism in business circles. Opinion is gaining strength that industry stands on firmer ground; but, despite the general improvement, continuance of the recovery trend is likely to be less rapid than some may hope. Operators who expect to remain in business will consider plans for cutting costs, if they have not already done so, to adapt themselves more quickly to market levels that are lower than those they consider normal. Plant rehabilitation is a step to this end. Normal replacements have been postponed in the last three years, and the fact has been overlooked that a neglected plant can lose money from many causes, such as inefficiency, breakdowns, lost time, and increases in the number of accidents to employees. Modernization is another step in the adaption of industry to changed conditions. Much of the equipment now in service is inefficient. Many improvements in design introduced since 1929 could be adopted with good results. Return of the investment involved can be expected in lower operating costs.

Failure to replace obsolete and worn-out equipment has been a factor in deepening the depression. In many instances it has lessened the operator's ability to sell his product in a highly competitive market. But the inactivity characteristic of the industries that produce machinery and plant facilities, because of the cessation of buying, has had a far worse effect, for these industries are the chief consumers of raw materials, including metal and mineral products. Their recovery will be stimulated by a widespread program of modernization.

If employees can be kept on the payroll at a time when production is not urgently needed, as is being done in many instances, common sense calls for providing work of a nature that will not add to unsold stocks. The division of the mining industry engaged in recovering gold is in an ideal position to put its plants in order. It offers an important market for equipment and supplies, and it is therefore in a position to contribute materially to a resumption of general business activity by encouraging the so-called capital-goods industries to add to their personnel and to increase their consumption of many raw materials.

Geophysical Prospecting Indicates Rand Extension

HISTORY has demonstrated on several occasions that a significant discovery of gold stimulates industrial activity and revives business. Although the annual production of gold has appreciated steadily in recent years, the discovery of no new and great deposit has been reported. Exhaustion of known reserves was expected to lead to a steady decline in world production as soon as the present spurt in output, stimulated by low prices for commodities and abundant labor, had spent itself. To counteract this unfavorable anticipation comes important news from the world's premier gold field, at Johannesburg, to the effect that Dr. R. Krahmann, of the University of Charlottenburg, has succeeded in demonstrating by aid of a magnetic variometer that the Main Reef Series is likely to extend to the west about forty miles beyond what has been considered its termination.

The prediction is based on the interrelationship known to exist between the gold reef and a reef of magnetite. Extension of the latter, which lies underneath the gold reef in areas being exploited, has been demonstrated by geophysical methods. A similar extension of the gold reef is hoped to be demonstrated by drilling exploration now under way. Indications are favorable, but the outcome must remain in doubt until positive sampling methods demonstrate the presence or absence of payable ore in commercial quantity. If the results are favorable, a vast tonnage of reserves may be quickly developed, sufficient to consider the extension as a new discovery that will have a profound and favorable influence on world economics.

Preparing for Economical And Efficient Production

DEPRESSION in the base-metal industries has one advantage: it provides opportunity for a survey of conditions in mine and plant, for the consummation of plans to insure continuous and satisfactory operation when normal consumption is resumed. Now is the time, while materials needed for a construction project are obtainable at low cost, when labor is abundant and zealous, to insure against future interruption to output or interference with technical result, by introducing reforms in design and refinements in equipment that cannot during normal times be effected in the thorough manner that is possible when production is temporarily suspended. Moreover, most mines and plants must be kept in condition, output or no output; and those who are engaged in such work now can be employed as well in some constructive capacity, whereby the cost of repairs or changes in design is proportionately reduced.

An example of executive foresight and engineering acumen is to be found in the article on "Concreting the Denn Shaft," by Mr. F. P. Brunel, on another page of this issue. While the property of the Shattuck Denn Mining Corporation, at Bisbee, Arizona, was shut down because of the low price of copper, opportunity was taken to put the Denn shaft in first-class condition, by concreting. The description omits nothing of importance. Complete engineering data are disclosed, providing information of value to other companies in a position to effect similar structural changes, and illustrating anew the generous recognition of a spirit of reciprocity and cooperation in the release of technical details that characterizes the policy of so many of the great mining companies, particularly in North America.

The relative costs of retimbering and of concreting were considered, and the more costly but more permanent method was adopted. The work was begun on Sept. 4, 1931, and finished on June 18, 1932, with an actual working time of nine months and fourteen days. Absence of interruption by production operations permitted a thoroughness and care in construction that otherwise would have been impracticable. Beyond business progressiveness and technical perspicuity, however, was the contribution made by the company to the improvement of local economic conditions. A monthly payroll of about $10,000 was maintained during the period, providing the work that offers the only satisfactory solution of the unemployment problem.

Statistical Comparison And Calendar Reform

SIMPLIFICATION of currency denominations by the adoption of the metric system has saved the Americas many millions of man-hours of useless labor and futile calculation, permitting mechanized computation and mechanical accuracy in result; but the leading English-speaking nations of the world still operate almost entirely under the handicap of an archaic system of weights and measures, against which absurdity the scientist and the engineer feebly protest, to little avail. Technology as well as business continues to use a calendar that makes impossible the adequate and logical recording of current performance or trend, because the official monthly units of time differ from each other with irritating irregularity. Comparison of the results in any one year of a month's operation with those of the previous month is feasible only in the case of August, which is preceded by a month with the same number of days; and the comparison is valid only when the number of working days is the same in each instance, which seldom occurs. Comparison is therefore usually made with the same month of the previous year, which gives no indication of current trend; or misconceptions and misunderstandings arise because of comparison with data covering the previous month, without drawing attention to the fact that the two periods differ in the number of days, and also in the number of working days.

The sooner the civilized world evades the self-imposed travail caused by this absurd system the better. A simplified calendar of thirteen months, each of four weeks of seven days, with an eighth-day "year Sunday" added to the last week of each year, is meeting with acceptance among those who scorn superstition and defy convention. The railway companies in England now use the new calendar for reporting operating revenues, and several large firms in the United States are also in favor of the thirteen-month year. The need was never more urgent for statistical data; but expense of collection is largely wasted if they cannot be used as reliable guides for comparative and indicative purposes. Operating costs and production, income and expenses—no

satisfactory check can be kept on these items unless trends are determinable by comparison of figures over equal periods.

The League of Nations is to make a recommendation in 1935 in respect to the universal adoption of the new calendar. In the meantime, progressive organizations and individuals will act wisely if they take advantage of the opportunity presented to break away from habit and precedent. The calendar in use is unsuited to the times; it is a costly and cumbersome heritage that should be discarded. Change is inevitable eventually. The Gregorian system does not even possess the one advantage that may be claimed for the use of complex weights-and-measures or currency systems: that work is thereby provided for those who otherwise might be unemployed. No amount of labor can make the archaic calendar now in universal use a help to increase group efficiency and economy or individual well-being.

Reducing Accidents By Improving Health

AN ACCIDENT, according to the dictionary, is an event that takes place without one's foreknowledge or expectation. But more attention should be paid to the fact that what is considered an accident by the victim may be viewed by an onlooker as part of a logical sequence of expected happenings.

The problem of preventing accident has been complicated by an illogical analysis of cause and effect. No jury of men and women of varied mental and physical qualifications could honestly agree as to what constitutes an accident or an "act of God." An inebriate who wrecks his automobile against a barricade is not entitled to protest that the obstruction was the cause of the disaster.

The majority of accidents, however, suggest a more subtle examination of antecedents. Of primary importance is the physical condition of the individual. If he be in good health, with no mental or constitutional defect, the hazard is slight. He may view an accident to another as inexcusable, but a medical examination of the victim will invariably reveal a condition that calls for remedial treatment. Threats or promises cannot prevent happenings that are the logical sequelae to ill health. The obligation rests on society and the state to teach the individual to seek the real cause, to forget incidentals to the happening, so that the basic incentive to the hazard may be removed. "Health First" would be a good slogan with which to draw attention to a frequent though generally unrecognized cause for the majority of accidents.

Low-Cost Production And Natural Advantages

MONOPOLY has been defined as the exclusive domination of the supply of a commodity or service in a market; or one that permits raising price above the free-competition level. But the tendency is common to accuse of monopolistic control an enterprise that produces an article at a lower cost than can its competitor.

Unmoral influence is implied in a threat made to "Break the World's Radium Monopoly." In 1913 the United States was producing almost all the world supply. Then came discovery of ore in the Belgian Congo. African radium appeared in 1922, at a price that inhibited production elsewhere; whereupon the Belgians were accused of monopolistic control, although they had no monopoly of world supplies, nor was the price raised above what would result under competitive conditions. They found high-grade ore, associated with other valuable minerals, and so could afford to reduce price.

The discovery of radium in Canada now deflects attention to the possibility of "breaking" what is termed a monopoly by the Belgians. However, the American industry was forced to the wall by price reduction in 1922; and the new find in Canada, if costs can be further reduced, may result in the transfer of the so-called monopoly to the Dominion.

Interpretation of low-cost production as essentially monopolistic ignores the trend in recent years toward efficiency and economy in manufacture, the quest for raw material of a richness permitting profitable exploitation. Those who are successful in search, who are favored by circumstance, reap the rewards.

Publicity and the Dust Abatement Problem

A GENERAL SURVEY of the mining industries will disclose conditions on many properties that deserve attention and suggest remedial action, especially in regard to hygiene and welfare. The dust problem underground is responding to a demand for reform that arose from statistical revelations indicating a neglect to realize the hazard to employees of the inhalation, in a confined space, of finely divided silica. The problem on the surface is equally serious but more open to attack. Its solution has been delayed in many instances because executives have considered it solely from the humanitarian point of view, whereas dust collection has been proved, in every instance where the work has been under competent and experienced direction, to justify initial and operating expense many times over. As Mr. George E. Lynch demonstrates in his article in this issue, on the design and operation of the dust plant at the crusher station of Roan Antelope, in Northern Rhodesia, the salvaged material usually pays a substantial profit on the investment; and the improved mill conditions tend to strengthen the morale of the staff, reduce labor turnover, lower the accident rate, and increase efficiency and output.

Mr. Lynch's article is timely and helpful and encouraging. Farsighted and humitarian executives, of which there are many, who realize that dust collection is good business as well as good engineering, are invited to take greater advantage of existing avenues of publicity in describing methods and results of combating this hazard. The outcome should shame into action those who are still apathetic or guilty of neglect.

Dust Control an Economy Measure

George E. Lynch

CONTROL and elimination of floating dust in crushing operations has been standard practice in all the larger copper plants of the United States and Canada for some years. Starting with the Phelps Dodge Corporation about ten years ago, the use of dust control, as distinct from older methods of dust collection, has so spread throughout the mining plants of the West that the old-style plant, with its steelwork piled high with dust, and the air constantly hazy with fine floating material, no longer exists.

In starting this work, we assumed, and correctly so, that a notable decrease of silicosis and other dust-caused ailments would be effected among the workers, and that the wear on machines and other equipment would be reduced. We did not foresee, however, that the salvaged fine material would, in every plant installed, pay a substantial profit on the investment, or that the cleaner mills would attract and hold a fine class of mill men, with consequent decrease of accidents, both to men and to machines, and increased efficiency and output from these causes. Altogether, this practice, started solely with the desire to safeguard the health of the workers, with little anticipation of financial return, has proved itself extremely desirable from many points of view, and the dust-control system is now as much a part of the modern mill as the crusher.

When the mill and concentrator for the Roan Antelope Copper Company was designed, a decision was made that the dust system should be provided for at the start, and not left until the mill was built, to be installed as best it could after all available space had been taken up with other machinery or piping. This required that the design of dust-control equipment be made directly from the drawings and that the equipment be installed and adjusted by the men at the plant, inasmuch as the distance was too great to permit a visit at any time by the engineer specializing in dust work. Up to that time, I had designed all dust-control plants directly in the field, and, although the work of making up piping and installing the machinery was done either by local contractors or by company men, I found that to return about the time of starting to correct and adjust the smaller troubles, so that the system could get off to a good start, was advisable. However, as many of the engineers at Luanshya had extended experience in the Southwest area of the United States, and were familiar with dust control, to leave it in their hands seemed entirely safe. Results justified this decision, reports to me stating that no trouble whatever was found in any part of the work.

The mill differs slightly from ordinary practice in that the various units are all in separate buildings, connected by inclined belt conveyors. As these buildings are some distance apart and on different levels, a single large dust plant would not be feasible, and a study was made to decide whether a number of separate systems would be necessary or whether the actual dust problem was not confined to one building alone, the dust in other departments being of insufficient volume or importance to justify the expense of a plant.

The buildings concerned are as follows:

1. Coarse-crushing department, which includes the ore bins and waste chutes at the head of the shaft and the two large gyratory crushers, with their grizzlies and chutes to conveyors.

2. Storage bins, which are served by conveyor No. 2. This conveyor takes ore from under the gyratories and delivers it to a tripper distributing conveyor, No. 3, at the top of the bins. Gates at the bottoms of bins deliver ore to belt conveyors Nos. 4a, 4b, 4c, 4d, and 4e, discharging at the top of the next building, which houses the screens and fine crushers.

3. Fine-crushing and screening department, containing the screens and Symons cone crushers. These crushers are in

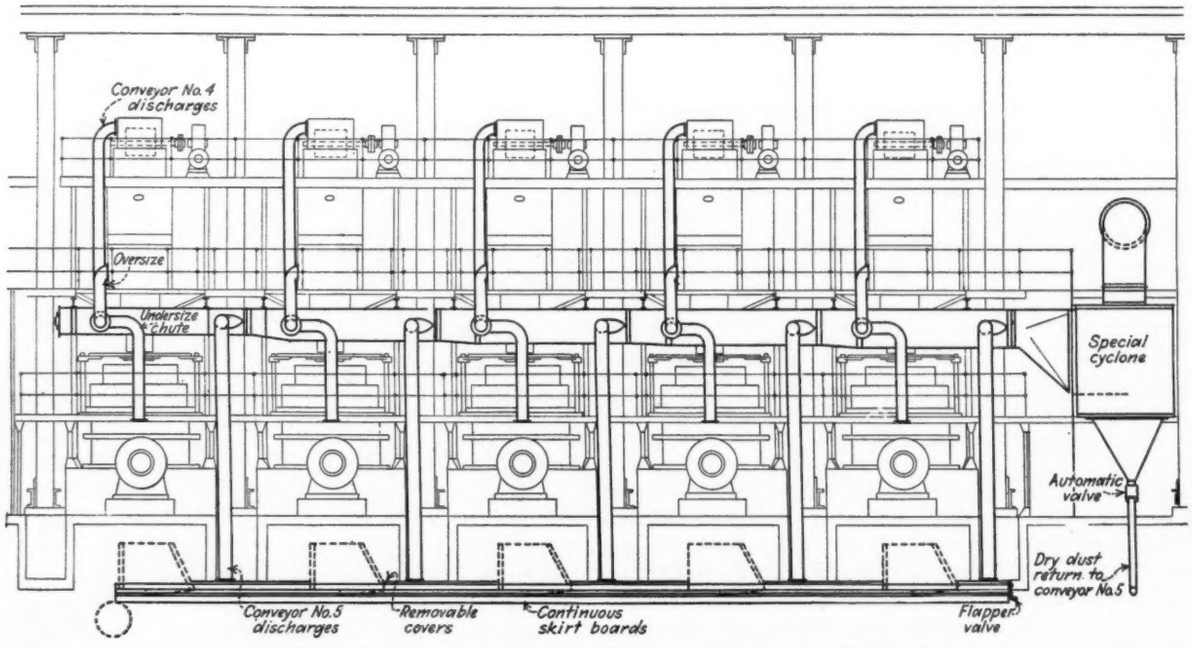


Fig. 1—Main header with suction connections and discharge to special cyclone

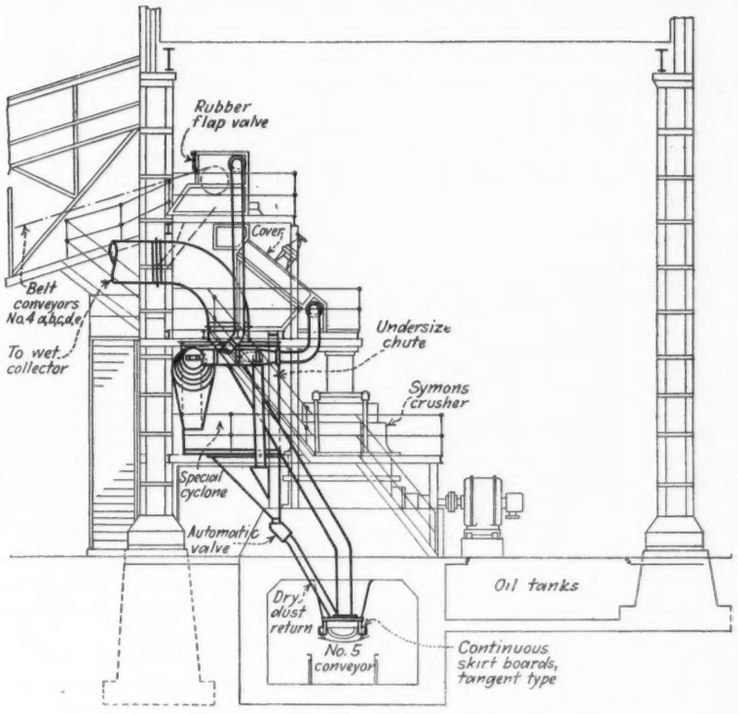


Fig. 2—End elevation showing special cyclone

open circuit, undersize from the screens being bypassed and joining the fine crushed product on belt No. 5, which runs in a tunnel under the Symons cones. This product is delivered to conveyor No. 6, which carries it to the top of the concentrator building, where it is distributed by a tripper to the bins.

Control of the dust at the coarse-crushing department would obviously be difficult—that is, if its importance justified the expense of a plant. The building and machines were open, and to inclose them in such a way as to keep the air volume to be handled down to an economical amount would be impossible. However, expectations were that the mine ore would be fairly wet and that dust from mine fines would not, therefore, be important, the only dry dust being that resulting from crushing. From experience with other plants of the type, this dust was known to be not especially bad, and, because only two or three men would be required to work near the machines, to oblige these men to wear respirators when actually in the dust was judged more economical.

The same considerations applied at the storage bins. Here, the difficulty appeared in the necessity of controlling the dust at the tripper. This has never been done with any economy, because the free area required by a tripper is so great that an immense volume of air must be handled to establish a satisfactory inward current at the openings. Here, also, only a few men had to work in the dust, and the expenditure of a few dollars for respirators was considered better than an investment of several thousand for a dust plant.

A real problem existed in the fine-crushing and screening department. The crushed ore and fines, partly dried by storage and handling, are discharged at the top of the plant, over the Hum-mer screens, and travel by gravity down through either the undersize chutes or the Symons cones, gathering in the tunnel below on conveyor No. 5. This department in every mill always provides a dust problem, which is increased in gravity by the fact that a number of men are constantly employed about the machines, their work being such that the use of respirators is awkward and undesirable. A decision was made, therefore, that a standard dust-control system should be installed here.

At the fine-ore bins, conditions similar to those at the storage bins existed, with the attendant difficulty of clearing about a tripper; but one or two men were required to work in the dust, and then only at intervals. A recommendation was made, therefore, that the top of the bins be shut off from the mill by light, dust-tight partitions, and that the men engaged in handling the trippers and attending to the conveyor be obliged to wear respirators when in the dusty atmosphere.

The dust-control system designed for the fine-crushing and screening plant follows the general principles of dust control, differing, of course, in the special applications which always necessitate that every dust plant be designed for its own mill alone. Briefly, a suction system maintains an inward current of air at all necessary free openings. This is accomplished by connections at proper points, with smaller pipes combining in a main header which delivers the collected dust to a dry collector or cyclone. The coarser dust is thrown down here, and the cleared air, along with the finer dust, then passes from the top of the cyclone to a wet collector or spray tank, where the fines are collected and delivered as a pulp containing about 6 per cent solids. The cleaned air then passes into the suction of an exhauster, which discharges it directly into the atmosphere.

The main header was installed above the Symons cones, directly under the longitudinal platform below the screens. This permitted a design of all pipes which would cause them to be self-cleaning, and also kept the piping out of the way of men and cranes. Only one connection on each screen has to be broken for repairs on the Symons cones, and these pipes are provided with flange couplings so that they can easily be removed and replaced. This feature is extremely important in any dust system, because crushers require frequent repairs and replacements, and, if dust piping is in the way, it is usually exposed to more or less damage, because repair work is always done in a hurry and mostly at night.

This main header, which is horizontal, passes to the end of the mill, where it discharges into a special form of dry collector. Two inlets are used on this collector, as the discharge of belt No. 5 onto No. 6 cannot easily be reached by a pipe from the header. The collected dust from the special collector discharges from an automatic valve into a sloping pipe, which delivers it on top of the ore on conveyor No. 5. Naturally, this dust immediately sinks into the ore, and no dust is visible after the belt has moved two or three feet. The whole quantity of collected dust rarely represents more than 2 per cent of the dust in the ore, so that its return to the flow, at any point, adds no difficulty which does not already exist at points beyond.

An important feature of the installation consists in the use of continuous skirt boards on conveyor No. 5. Usually, where a number of machines discharge onto a single conveyor, the custom is to provide separate boxes, each with its own short skirt boards and with open spaces between. This method makes almost impossible prevention of the escape of dust at the ends of these separate boxes, and also assures avoidance of spilling by the plowing off of excess load from some previous chute when this material reaches the next box. Here, all the suction connections open into a common long box, built over the belt and provided with continuous rubber skirt boards which bear at a tangent on the belt. The ends of the box are closed by flappers or curtains of rubber belt, and the tangent type of skirt board effectively eliminates spill. The suction of

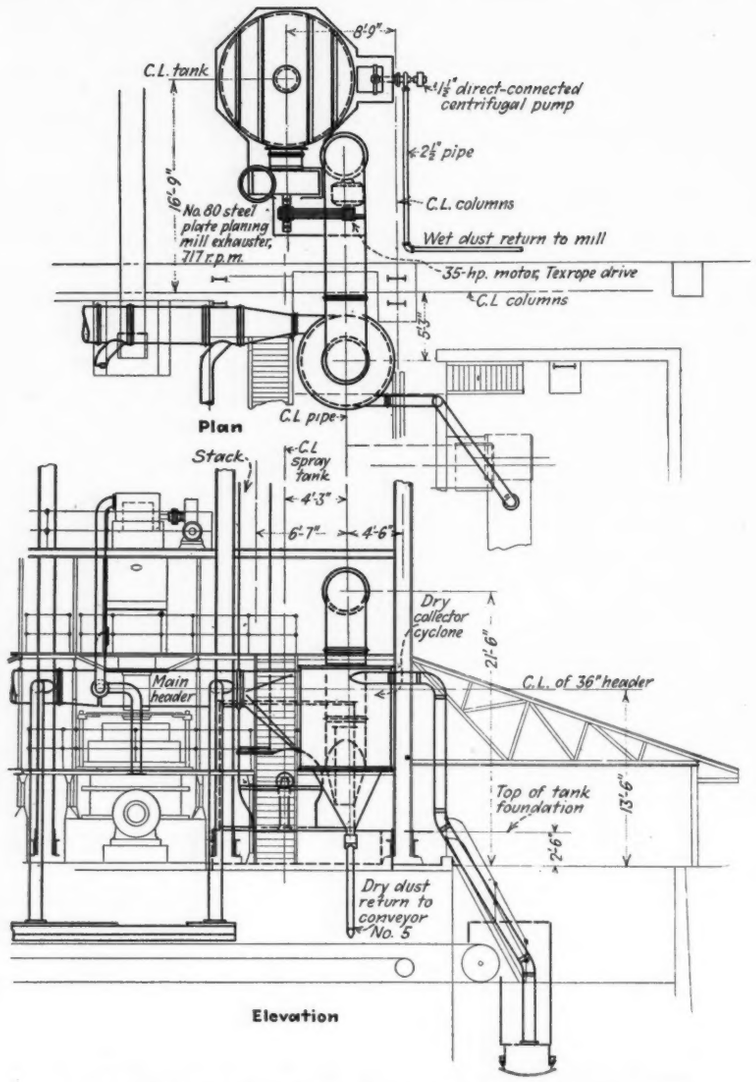


Fig. 3—General assembly showing cyclone, wet collector, and fan

the pipes, distributed along the entire length of the box, causes air to move inwardly at every opening, and thus neither dust nor ore can escape into the tunnel. Any mill man who has had to shovel up two or three tons of dust and spill in a tunnel at the end of every shift will appreciate the advantages of this method.

The accompanying illustrations show the general layout and assembly of the dust-control system. Fig. 1 presents a front elevation of the main header, with its suction connections and discharge to the special cyclone; Fig. 2 shows an elevation from the end opposite the cyclone, and indicates the clearance and passageway left for the men. A plan and elevation of the cyclone, wet collector, and fan are presented in Fig. 3.

Inasmuch as the fine-crushing plant is about 5 ft. lower than the concentrating mill, where the collected fine dust must be delivered as pulp, necessity existed for the provision of the small, direct-connected centrifugal pump at the discharge of the wet collector, to elevate and convey this pulp to a point where it could be distributed to the classifiers, in which it is generally returned to the mill flow. Fortunately, upon operation of the dust system, the discovery was made that the first or dry collector gave an entirely satisfactory clearance of solids, and that the remaining fine dust could be discharged into the air without causing a nuisance. This enabled the wet collector and its pump to be omitted, and avoided the complication the equipment caused. However, this result is seldom attained in American mines, and I did not feel safe in omitting the wet collector in the original design, as mine moisture might vary greatly, and a single cyclone rarely gives sufficient clearance to permit discharging directly into the air near the mill.

The piping, tanks, and other machinery for this plant were made in England and shipped to South Africa in a more or less knocked-down form. As no direct information was available regarding clearances on the railway from the coast to Luanshya, a decision was made to design the large parts of the equipment in sectionalized form, so that the tanks, in particular, could be nested for shipment, and then assembled in place by riveting, bolting, or welding. The use of sectionalized tanks and cyclones is frequently desirable, even when no shipping difficulties exist, because these parts are usually installed inside buildings, and the sectionalizing avoids the necessity of cutting openings in walls to admit them. The exhauster is known in the United States as a No. 80, steel plate, planing-mill fan. This type of fan has flat blades, of heavy steel plate, and may easily be repaired at the mill when the blades are eroded or otherwise worn. It runs at 717 r.p.m., and is driven by a 35-hp. motor through a Texrope drive.

Reports on the operation of this plant, after some months' use, are very satisfactory. The air has been entirely free from dust, and no clogging or filling has occurred in any of the pipes. This is especially gratifying in a plant designed from drawings alone, and erected on the other side of the world. Naturally, I used my own best efforts, but I am forced to attribute a large part of the success to my good friends at Luanshya.

Waste dump at Alaska Juneau

Limitations of Economic Geology

Charles A. Porter

IN THE FIELD of metal mining, perhaps no other subject has so vague a value as economic geology. Opinion differs greatly as to its worth. To the so-called practical man, geology is not infrequently an object of contempt. On the other hand, most large mining organizations maintain permanent geological staffs, or at least make no move in exploratory work without consulting their geologist. Between these two extremes all shades of opinion may be found. That geology has not completely filled the expectations of a quarter of a century ago can hardly be gainsaid. Nevertheless, in all but the simplest ore occurrences it seldom fails to have an appreciable value; and in the more complicated conditions its worth is likely to be increased to the point where it is the deciding factor between success and failure.

What are the factors that determine the value of economic geology, and what are the elements that set bounds to its use? Endless pages deal with the geology of ore deposits, and in them, often with great detail, peculiarities of different ore occurrences are described. Much of this information is of value to the practical miner, more is of doubtful worth, and still more is obviously useless. But through this mass of literature one is surprised that so little endeavor, if indeed any at all, has been made to analyze the difficulties inherent in the application of the science. The more advantageous features have been emphasized, if not exaggerated, while the difficulties have been passed over, leaving them to be learned, if at all, by bitter and expensive experience.

Although to attempt to establish limits to such a complicated and indefinite subject may seem too fanciful, certain fundamentals are amenable to generalization. Through such generalizations I hope to bring out clearly the nature of some important limitations commonly met in applying the science. To establish such limitations is desirable. Not only to the geologist is it advantageous but to those who utilize him as well. For out of such an improved understanding must result a curbing of ungrounded and unreasonable expectations on the client's part and of the pretensions on the part of the geologist that are so prone to attach themselves to the professions in general.

At the outset, we must revert to first principles. We must reconstruct before the mind's eye a page from the obvious. We must consciously realize that fracturing is a prime essential to the formation of mineral deposits. It is fracturing, pre-mineral fracturing, that opens the way for the flow of solutions. Such fracturing is patently the *sine qua non* of ore deposits; and may precede not only the deposits, but the intrusions also that have occupied such a large place in the geological literature of the present century. It may seem singular that more curiosity has not been displayed as to whether the fractures or the eruptive were first. I discussed this point some years ago and have reverted to it at times since then, and will therefore pass it by, as it is not strictly essential to the present discussion.

Regardless of its possible priority to the accompanying intrusives, in this fundamental fracturing we have a key to the primary and preponderating difficulty in applying geology to ore discovery. We all know that, even in well-mineralized areas, some fractures carry ore and many do not. From this one readily sees that it is of the highest importance to tell beforehand which fractures are ore-bearing and which are not. No geological principle will enable us to do this, and geology gives us no aid on this point. From his experience, every miner knows that ore-bearing fractures are never uniformly mineralized. In such fractures the ores may occur in pockets, as shoots, or as bodies irregular both with respect to their strike and their dip; and it is a good vein indeed that is not more than two-thirds barren. Moreover, a fracture may serve merely as a channel through which metallizing solutions have flowed, leaving nothing but a thin seam of indifferent ore as evidence of their passage, and carrying the essential mineral to some favored spot, as, for instance, to a mass of limestone, or a bedding plane where the deposit may be formed away from the feeder itself. Or, possibly, the mineral was laid down high up in some ancient formation, which through erosion has been destroyed along with what was once an ore deposit.

Obviously, fractures in the earth's crust do not occur without cause. What these causes are is still a matter of speculation. The complex fracturing so common in mineral districts is nevertheless usually attributed, consciously or unconsciously, to igneous action. Possibly this source of energy will, in the end, prove sufficient for all purposes. But, assuming that such is the case, we have no quantitative standards as to how much energy there was, nor where it was applied; nor do we know the manner of its application. In short, our precise knowledge of those great upheavals that give rise to the ruptured conditions accompanying ore are not only general but also very vague.

When we find ourselves involved in such uncertainties in dealing with the broad general phases of the subject, we must, when we descend to details and attempt to forecast the actual occurrence of ore, expect to encounter still more formidable difficulties. At this point we are likely to find ourselves in a sea of troubles, inasmuch as ore, when it occurs at all, occurs usually in unpredictable quantities, and in just as unpredictable places. We know that metals are carried to their points of deposition by solutions that pass through fractures. Nevertheless, we have no criterion by which we shall judge beforehand which solutions have been mineral-bearing; which channels have carried such mineral-bearing solutions; or whether, before ore has been found, there have been any ore-bearing solutions at all or suitable fissures to serve as ducts for their conveyance toward the surface. In fact, we do not know how to distinguish between a suitable and an unsuitable fissure.

After ore has been found, mineral-bearing solutions may be inferred, and, not infrequently, the channel or channels through which they flowed may be determined. But, in such cases, the first step has been either the result of chance or of the empirical efforts of the prospector. This is the course that events usually take in the early stages of ore discovery. Seldom does scientific geology become of much importance until ore has been exposed and something of its character and habit determined by the less refined methods.

Thus far in this discussion, only the primary aspect—that is, the supergene phase—of ore deposition has been considered. In the consideration of enrichment due to surface factors, geology has a much greater value in the early stages of exploratory work. In estimating the probable value of many copper deposits, conditions are reversed, and the geologist may profit but little from the work of the prospector. Here the truly scientific geologist, who has had adequate experience with outcrops of copper ores especially, is frequently able to make forecasts of accuracy sufficient to meet the hard test of commercial value. These two phases of geology—namely, the primary and secondary (that is, the supergene and the hypogene)—probably stand as representing the two extremes in the value of economic geology.

Another phase that often attains a

high practical value is the solving of complicated problems in faulting. Here scientific geology may accomplish much where empirical methods have little better to offer than guesswork. At this point the empiricist is often entirely unaware of facts of great practical importance. To this may also be added the vague influence that the scientific study and comparison of different deposits exert, which so often gives significance to much that passes unnoticed under the pragmatic eye.

From the foregoing one may see that the value of geology is highly variable and in its application there is a large region where science and empiricism may meet and neither be dogmatic. The great maxim of the empiricist is, "Follow the ore." In practice this frequently begins with tracing float to an outcrop, perhaps up a long gulch and a hillside, and panning dirt to aid in locating a vein for which one may have to trench and then following the vein itself to determine the quality, quantity, and character of ore. Again, as a mine or part of a mine approaches exhaustion, the principle may be reverted to and seams or barren portions of veins may be followed in the hope of picking up additional orebodies that give no clue to their existence beyond the presence of the fracture followed.

Evidently, economic geology, even as a science, is still imperfect, and when it is applied to the art of finding ore, its imperfections become magnified. Very often, it can do no better than to estimate probabilities, and indeed not even that at times. On the other hand, seldom is a stage reached where the science is entirely useless. Frequently, the purely practical man wastes his efforts upon that which, to the geologist, is obviously hopeless. Herein lies a value that alone justifies geology—its power to save waste. An objector may say that geologists have passed adversely upon this or that great mineral district. No doubt that has happened, but such statements are easily invented, as was the one current twenty or twenty-five years ago that a then very prominent mining engineer had "turned down" Goldfield, Nevada. As a matter of fact, that engineer was never on the ground until several years after the district had become famous.

Moreover, all is not a geologist that glitters. The apprenticeship is necessarily arduous and prolonged. Much of the study of geology is scientific in the extreme, and much must be learned in the field. As has already been stated, there is a large region where science and empiricism meet, and there we find a world of compromises, a world of modifications, a world of elucidations—observation checking theory and theory explaining observation. Here is little room for rote learning, and the mental effort necessary to solve the problem is frequently arduous in the extreme. The task is one not for the conventional specialist, but for a specialist familiar with the best that science can afford.

Applied geology is not a subject that he who runs may read, because such a one can hardly form a proper picture either of its possibilities or of its limitations.

■

Old Automobile Makes Efficient Hoist

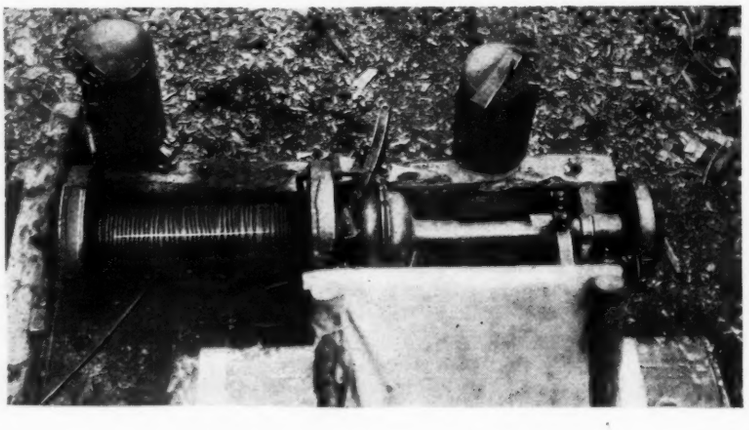
THE hoist shown in the accompanying illustration was made from an old Chevrolet touring car by H. B. Reser, for the purpose of sinking a shaft on his lease in the Hog Heaven mining district, near Kalispell, Mont. Construction of the hoist is simple, and may be duplicated by anyone mechanically inclined who can secure an old automobile with an engine in good running order.

In making the hoist, first strip the body from the car, take off the wheels, and then anchor the chassis to the ground in the correct position in relation to the headframe. Next, cut off the housing of the rear right axle about ¾ in. from the outside end of the differential carrier bearing, thus allowing room for a felt packing ring within the axle housing to retain the grease in the differential housing. The differential is

■

locked by cutting keyways in the ends of the axles, and by keying the two shafts together with a steel key. The spider gear is then assembled, and the spider gear housing is poured full of molten lead, thus making in effect a solid axle. An extra rear hub for the same make of car as that used is obtained and bored with taper hole, until it will drive over the axle. After being bored out, sufficient keyway will remain in the inner end of the hub to key it to the axle about 2 in. from the felt packing ring, a suitable keyway having first been cut in the axle. Another hub is bored out and keyed in the same manner to the outer end of the axle, leaving

■



enough of the axle protruding beyond the hub for a regular bearing. Two large brake drums are bolted, open ends outward, to the hubs. These form the ends of the rope drum and the brakes, using the regular bands and linings.

Before putting the outer hub and drum on the axle, a 22-in. section of 6-in. pipe or well casing is slipped over the axle. This is centered between the brake drums, and brazed or welded to them, thus forming the winding drum of the hoist. The bearing for the outer end of the shaft in the installation under discussion was fitted into a timber, as shown in the accompanying illustration.

The improvised hoist can be operated by anyone who can drive an automobile. The hoist described lifted a bucket carrying 300 lb. of rock when operating in "second" gear; a much heavier load could be lifted by operating in "low" gear. In lowering men and timbers, the machine was placed in "reverse." A very steady control was obtained with the foot brake. In lowering the empty bucket, the machine was placed in "neutral," the descent being controlled by the foot brake. The ⅜-in. wire rope used for hoisting purposes was wound on the drum without difficulty.

The old Chevrolet cost $15, and the alterations and installation cost $25. Thus, the total cost for a small, serviceable, and efficient hoist was $40. Moreover, in this particular installation, the flywheel of the engine was belted to a small fan that served to exhaust powder fumes from the shaft after blasting.

Summary of the Cost of Mining

Morris J. Elsing

THE United States Bureau of Mines reports that in 1929 a total of 98,380,455 tons of ore was mined in the United States by underground methods from mines that have an annual production of more than $100,000. This tonnage, segregated by methods, is given in Table I.

Table I—United States Production by Methods in 1929

Method	Tons	Per Cen
Open stoping	35,186,673	36
Block caving	18,576,196	19
Top-slicing	15,904,288	16
Shrinkage	11,034,148	11
Square-set	7,819,062	8
Sublevel caving	7,303,976	7
Cut-and-fill	2,556,112	3
Total	98,380,455	100

In previous issues of *Engineering and Mining Journal,* I have made an attempt to establish an average figure that should be representative of the cost per ton for mining by each of the methods listed in Table I. The most important factor in judging the accuracy of average costs is the proportion of the whole tonnage they represent. Naturally, if the cost per ton of the total tonnage mined by any one method were available, an accurate average could be determined. The total costs per ton of mining 36,000,000 tons by underground methods have been presented. All this tonnage was not mined in 1929, but it was mined in years sufficiently close to that year to be representative. Incomplete costs of several million tons more have also been presented. Table II shows the approximate tonnage for which costs have been given for each method, and the total tonnage mined by the different methods in 1929.

Table II gives costs for 36 per cent of the total tonnage mined in 1929. The block-caving costs are most representative, with 90 per cent of the total. Including the incomplete Morenci costs, the costs per ton by block caving are almost complete. Top-slicing is most poorly represented, with costs for only 6 per cent of the tonnage mined by this method. Costs for a large additional but unknown tonnage have also been considered in the calculation of average costs. In addition to this, in the data for many mines, averages have been checked against costs for many years back, to make sure that the costs used in determining the average are not unusual or exceptional.

Table III shows arithmetical averages of the costs of mining by the different methods for 81 mines, representing a substantially larger production than 36,000,000 tons of ore. The tonnages produced from the different mines have not entered the calculation of averages. A certain amount of discretion has been used in making these averages. For example, one mine, using the square-set method, reports a cost for one month of $9.50 per ton. Because of exceptionally difficult ground conditions, and the short period of time covered, this cost was eliminated in determining the average cost of square-set mining. In the data on the cost of shrinkage mining, the Alaska Juneau figure of 29c. per ton has not been included in the average of the cost for this method. This cost is so remarkably low that it should not be included in establishing an average for the ordinary conditions of shrinkage mining.

Total cost of stoping and mining per ton is an average of these costs for the different mines. The sum of the individual items for any one method does not agree with these totals. Classification of costs includes the six primary methods of underground mining and the sublevel caving variation of caving and the Iron Country variation of top-slicing. The cost of the inclined variation of any mining method has been averaged with other costs of the method.

The inclined variations of the cut-and-fill, top-slice and square-set methods, where applicable, result in lower costs per ton than by the horizontal method. Colorada mine, Cananea, and the United Verde, Jerome, report costs no lower for the inclined than for the flat cut-and-fill method. This is due to local conditions. Nacozari reports a saving of about 10 per cent. No comparative data are published by Calumet & Arizona—the most important user of the inclined method. The saving of the inclined over the horizontal method probably varies from 10 to 15 per cent of the total stoping cost.

Table II—Tonnage Represented by Costs

Method	Approximate Tons Represented by Costs Given	Tons Mined 1929	Per Cent
Open stoping	6,700,000	35,186,673	19
Block caving	16,650,000	18,576,196	90
Top-slicing	1,000,000	15,904,288	6
Shrinkage	6,500,000	11,034,148	59
Square-set	2,000,000	7,819,062	26
Sublevel caving	1,000,000	7,303,976	14
Cut-and-fill	2,000,000	2,556,112	78
	35,850,000	98,380,455	36

Three mines that have used the inclined top-slice method report an average reduction in the stoping cost of 17 per cent over the costs shown by the horizontal method.

No comparative figures are available for inclined or rill square-setting and the horizontal method. The saving, mainly in the cost of mucking waste and ore, probably amounts to from 10 to 15 per cent of the stoping cost.

Table IV shows the relative costs per ton of each item of each method by the numbers 1 to 8, 1 representing the lowest cost and 8 the highest. Table V shows the percentage relationship that each item bears to the total cost of mining for each method.

Many interesting comparisons can be made from Tables III, IV and V.

Table III—Average of Mining Costs per Ton by Various Methods

	Block Caving		Open Stoping		Iron Country Top-Slicing		Iron Country Sublevel Caving		Western Top-Slicing		Shrinkage		Cut-and-Fill		Square-Setting	
	No. Mines	Cost per Ton	No. Mines	Cost per Ton	No. Mines	Cost per Ton	No. Mines	Cost per Ton	No. Mines	Cost per Ton	No. Mines	Cost per Ton	No. Mines	Cost per Ton	No. Mines	Cost per Ton
Stoping																
Labor	3	$0.14	22	$0.31	3	$0.36	2	$0.36	8	$0.62	11	$0.75	11	$1.01	13	$1.44
Superintendence	2	0.02	18	0.03	3	0.03	2	0.03	2	0.08	10	0.09	5	0.11	11	0.11
Explosives	3	0.03	22	0.09	3	0.06	2	0.06	6	0.08	10	0.24	11	0.16	12	0.11
Timber	2	0.03	22	0.00	3	0.09	2	0.06	6	0.24	8	0.09	11	0.22	12	0.36
Air, drills, supplies	2	0.06	15	0.08	3	0.05	2	0.04	4	0.11	9	0.20	9	0.21	13	0.15
Power	1	0.01	7	0.03	3	0.02	..	.....	1	0.03	7	0.12	4	0.06	5	0.07
Miscellaneous	1	0.01	13	0.04	3	0.05	2	0.02	8	0.14	9	0.13	10	0.21	13	0.19
Total stoping	5	$0.23	22	$0.50	3	$0.65	2	$0.56	10	$1.15	12	$1.46	13	$1.84	14	$2.28
Development	5	0.17	13	0.22	3	0.09	2	0.25	3	0.27	9	0.67	12	0.76	9	0.68
Transportation	5	0.12	20	0.27	3	0.14	2	0.33	3	0.15	10	0.43	12	0.42	8	0.44
General	4	0.13	17	0.10	3	0.16	2	0.10	3	0.14	6	0.13	11	0.41	8	0.40
Miscellaneous	4	0.09	16	0.08	3	0.04	1	0.06	3	0.11	6	0.12	9	0.20	10	0.61
Total mining	4	$0.75	23	$1.04	3	$1.07	2	$1.28	3	$1.79	13	$2.41	13	$3.68	10	$4.13

This article is one of a series on the cost of metal mining in the United States.

Table IV—Relationship of Costs per Ton

	Block Caving	Open Stoping	Iron Country Top-Slicing	Iron Country Sublevel Caving	Western Top-Slicing	Shrinkage	Cut-and-Fill	Square-Setting
Stoping								
Labor	1	2	3	3	4	5	6	7
Superintendence	1	2	2	2	3	4	5	5
Explosives	1	4	2	2	3	7	6	5
Timber	2	1	4	3	6	4	5	7
Air, drills	3	4	2	1	5	7	8	6
Power	1	3	2	..	3	6	4	5
Miscellaneous	1	3	4	2	6	5	8	7
Total Stoping	1	2	4	3	5	6	7	8
Development	2	3	1	4	5	6	8	7
Transportation	1	4	2	5	3	7	6	8
General	2	1	4	1	3	2	6	5
Miscellaneous	4	3	1	2	5	6	7	8
Total mining	1	2	3	4	5	6	7	8

Agreement is general that block caving is the lowest-cost method, and square-setting the highest, but there is some disagreement as to how the other methods rank. I admit that variation occurs in the costs of any one method, but believe that, given average conditions for its application, costs will be more or less uniformly the same. The processes involved under a given set of conditions are identical. Policies of operation affect costs per ton much more than do the processes of operation.

Table IV gives the general relationship of the total cost per ton by each method. If the cost of each operation for any one method is estimated, and a theoretical cost set up for each method, the same relationship of costs per ton may be expected, as shown in Table IV. For example, the cut-and-fill method and square-setting have much in common. Both require filling; the stope development for both is almost the same; the method of breaking ore does not vary greatly. As usually applied, the cut-and-fill method necessitates a considerable amount of square-setting for the removal of crown and level pillars, and pillars between stopes are often removed by square-setting. Under some conditions, 30 per cent of the ore from a cut-and-fill stope comes from its square-set auxiliary. The timber cost per ton, as shown in Table III, ranks the third highest. The two methods differ mainly in that with square-setting timber is used as a temporary support. It is, therefore, logical that the costs of the cut-and-fill method should be high.

The most difficult of all methods to rate is Western top-slicing, because the method is now infrequently used on a large scale, and total costs per ton are not available. The average as shown in Table III is probably low, and my estimate is given as $2.15 per ton for the total cost of mining, and $1.15 per ton for stoping. Under favorable conditions, top-slicing should give lower costs than these. Such a conclusion is well justified by the results obtained at the Miami mine, in Arizona, from 1915 to 1918. This mine has produced the greatest tonnage (4,500,000 tons) of any top-slice operation in the Western United States. The costs given show $0.55 per ton for stoping, and a total cost of mining of $0.88 per ton. Average stoping cost for ten Western mines in years varying from 1916 to 1930 was $1.15 per ton. The total mining cost of these mines is not available. Average stoping cost in 1929 at the Old Dominion, Cananea, and Nacozari was $1.65. At Cananea, top-slicing has produced a tonnage in excess of the amount produced at Miami. The average cost of this tonnage was probably less than $2.25 per ton. A recent published cost from there is $2.19 per ton.

The Iron Country of Michigan in 1928 and 1929 produced 1,000,000 tons, with an average stoping cost of $0.65 per ton, and a total mining cost of $1.07. These costs are lower than the estimated average of top-slicing for the Western United States. The principles involved are the same, but operating conditions are less favorable in the West. An inferior grade of timber is used in the Iron Country, at a cost of one-third of the timber cost for the Western United States. In addition to this, the ore is broken at a much lower cost, and other favorable conditions exists which result in lower costs than can usually be obtained in the Western United States.

Table IV indicates that the ratio of the stoping labor cost per ton for each method corresponds with the ratio of the total mining cost for that method. Superintendence also corresponds closely to this ratio. A rather close relationship exists between the item "explosives per ton" and the item "air, drills and drill steel per ton." Hard ground generally increases the cost per ton for these items, and, therefore, if one is high, the other may be expected to be high. The hardest ore and waste rock are found in shrinkage mines. Therefore shrinkage would naturally show the highest explosive cost per ton and the second highest air, drill, and drill-steel expense. The lowest cost for explosives is shown for block caving, because most of the ore is won without the use of explosives. Ore mined in the Iron Country by top-slicing and sublevel caving is soft, and drilling is done with auger drills, which result in the lowest drill cost and the second lowest explosive cost per ton. The average ground mined by square-setting is softer than that mined by cut-and-fill stoping. This condition is reflected in a lower cost per ton in square-setting, for explosives, and drilling expense. Ore mined by open stoping is usually medium hard, but blasting is done to greater advantage than, for example, in square-setting, which accounts for a lower cost per ton for explosives and drill expense.

The timber costs per ton agree almost exactly with what is logically expected by the necessities of each method. That shrinkage and top-slicing in the Iron Country show identical costs per ton is surprising, but this can be explained by the fact that the timber used in the Iron Country is of inferior grade and is obtained at low cost.

No special significance is shown by the costs per ton for power and miscellaneous expense. They agree more or less with what should be expected, in that costs per ton for block caving are the lowest, and for cut-and-fill and square-setting, the highest.

The cost of development per ton is usually a variable factor not only for each method but also over a period for each mine. In most mining operations development comprises several distinctly different kinds of work. It includes work of a purely prospecting nature and work that is essentially stope development. Then come haulage development and shaft sinking. Some work is usually done for the improvement of ventilation. To keep these items separate is desirable, but this is impracticable except in certain specific examples of the larger extraordinary expenses. Almost all drifting, raising, shaft sinking, and diamond drilling costs are usually combined under the heading of development. Stope development is more or less a constant factor for each method, but all other development, a variable factor, depends upon the amount of ore reserves, the price of the metals as effecting expansion, the age of the mine, and other factors. In spite of all of the variables that go to make up development, the averages for the different methods are surprisingly logical. The average for the Iron Country top-slicing is probably lower than it should be. Western top-slicing is also a little low, and perhaps square-set development should be slightly higher than cut-and-fill. To expect that development costs should be roughly in the same ratio as the total cost of mining would be logical.

Table V—Percentage of Total Costs per Ton

	Block Caving	Open Stoping	Iron Country Top-Slicing	Iron Country Sublevel Caving	Western Top-Slicing	Shrinkage	Cut-and-Fill	Square-Setting
Stoping								
Labor	18.7	29.8	33.6	28.1	34.6	31.1	27.4	34.9
Superintendence	2.7	2.9	2.8	2.3	4.4	3.7	3.0	2.7
Explosives	4.0	8.7	5.6	4.7	4.4	10.0	4.4	2.7
Timber	4.0	0.0	8.4	4.7	13.2	3.7	6.0	9.0
Air drills	8.0	7.7	4.7	3.1	6.1	8.3	5.7	3.6
Power	1.3	2.9	1.9	..	1.7	5.0	1.6	1.7
Miscellaneous	1.3	3.8	4.7	1.6	7.8	5.4	5.7	4.6
Total stoping	30.7	48.1	60.7	43.8	64.2	60.6	50.0	55.2
Development	22.7	21.2	8.4	19.5	15.1	27.8	20.6	16.5
Transportation	16.0	26.0	13.1	25.8	8.4	17.8	11.4	10.7
General	17.3	9.6	14.9	7.8	7.8	5.4	11.1	9.7
Miscellaneous	12.0	7.7	3.7	4.7	6.1	5.0	5.5	14.8

Transportation costs for block caving are the lowest, because the largest tonnages are trammed and hoisted at each property. Mine cars and hoist skips are unusually large, permitting most economical tramming and hoisting, with automatic dumping. Open stoping, which ranks in second place as regards the total stoping and mining costs, holds fourth place in transportation costs. Transportation costs for cut-and-fill, shrinkage, and square-setting are almost identical. In general, transportation costs for cut-and-fill and square-setting should be similar, because the usual ground conditions that govern the adoption of the methods are similar. The deciding factor in the choice between the two methods depends on whether or not the ore and waste walls require the temporary support afforded by square-set timbers. The interrupted outputs of both types of stopes, because of the need for filling, the size of stopes, the spacing of chutes, and the method of haulage development, are similar for both methods. Similar transportation costs are, therefore, the natural result. In the application of shrinkage, conditions are different from cut-and-fill and square-setting. Some of the transportation factors are favorable and others unfavorable, with the result that the cost is similar to that of the two other methods.

The item of general and miscellaneous expense is likely to be a "catch-all" for charges not directly attributable to any of the other cost segregations. At least several of these have no direct connection with the mining method employed. For example, the presence of water might add a substantial cost per ton for pumping, which would be independent of the method of mining, and, therefore, because of numerous other similar charges, little uniformity or relationship should exist as between methods. More variation probably exists in the charges made under the heading of miscellaneous expense than with any segregation. In most records this item includes surface expense directly chargeable to mining. Considerable variation occurs in making such a distribution, and, therefore, uniformity as to methods cannot be expected. One reason that this item is so high for block caving is that prepaid development has been charged to this account. In spite of the diversity of charges in these accounts, the costs show the largest items for cut-and-fill and square-setting.

The data as presented in Table III comprise arithmetical averages of the costs as reported by the various mining companies. I believe that in some operations for which data are given the average cost is more accurately established than in others. Based on many unpublished and incomplete costs, Table VI has been prepared to show a summary of average costs, with a few changes to make it somewhat more accurate. Usually, the costs in Table VI follow closely the actual results at the various mines. Modifications have been made only where the arithmetical averages needed adjustment on the basis of the unpublished and incomplete costs, and where an exceptionally low or high cost has had an undue effect on the average.

Briefly commenting on some of the most interesting of the costs in Table VI, the total costs of mining by block caving and shrinkage show the greatest variations. Block caving costs clearly indicate that although as low a cost as 40c. per ton can be made, costs may be as much as $1.25 per ton. This variation is due almost entirely to ground conditions. If these are unfavorable, a high cost is inevitable. For average conditions a cost of about $0.75 should be obtained.

Shrinkage mining shows an even greater variation than block caving. Alaska Juneau has mined 36,000,000 tons at a total mining cost of 26c. per ton. At present the cost is about 29c. per ton. The Beatson and Alaska Gastineau costs were less than 80c. per ton. Cost of mining veins of different width varies from $1.75 to $3 per ton. The latter cost is ten times the cost of that at Alaska Juneau.

The total mining cost by square-setting varies from about $3.25 to $4.25, where ground conditions are favorable. With unfavorable ground conditions the total cost may be as high as $9 per ton. The average for all conditions is probably about $4.50 per ton.

Table VI indicates that in almost every instance the total cost of mining is twice as much as the cost of stoping. In caving and glory-hole mining this ratio is about three to one. On the other hand, the total cost of mining by open stoping in the Tri-State district is only 50 per cent higher than the stoping cost. These differences are due to the varying complexities of the different methods. Open stoping, as practiced in the Tri-State district, is the simplest of all methods. This method also has an added advantage of tying up the least capital in drifts and raises and in broken ore.

In writing this series of articles on the cost of mining in the United States two objectives have been kept in mind: to show that considerable uniformity exists in costs for each method, and that variations are due to local ground conditions; and, further, that efficiency of management is a more important factor in determining unit costs than the wage scale. Each operation in mining is more or less standard, and many of the costs are uniform in various parts of the country, regardless of the wage scale. The unit costs of mucking, drifting, raising, and shaft sinking are further proofs of this.

The other objective is to establish or determine what the average unit costs are for each method of mining under average ground and location conditions.

Study of cost data is beneficial because, after all, perhaps the greatest advances are made by the imitation of the latest and best practice of some other mine, still further modified and improved to fit local conditions. The best practice is often the result of the necessity of maintaining the life of the enterprise rather than the winning of greater profits. Rich high-grade ore is not an incentive for low costs. When the grade of the ore drops to the point of vanishing profits, drastic action becomes necessary to cut costs.

Great differences are noticeable in the policy of the different companies in regard to making their costs public. Some companies give complete details in their annual reports. Others freely allow publication of their costs in the technical press. The policy of other companies is marked by great secretiveness in regard to costs. Practices and methods are freely described, but publication of costs is taboo. This policy is subject to argument. Certainly the frank, open policy of making costs public is beneficial to the general advancement of the industry; and, likewise, the greatest number of shareholders are benefited.

Table VI—Summary of Mining Costs per Ton

	Total Stoping	Total Mining	Estimated Average Total Mining
1. Caving			
(a) Block	$0.15–$0.35	$0.40–$1.25	$0.75
(b) Sublevel	0.55	1.25	1.25
2. Open Stoping			
(a) Tri-State	0.60	0.90	
(b) Michigan copper	0.50	1.20	
(c) Iron mines	0.50	1.20	
(d) Glory hole	0.35	1.00	
Average			1.10
3. Top-slicing			
(a) Michigan iron	0.65	1.10	1.10
(b) Western U. S.	1.15	2.15	2.15
4. Shrinkage			
(a) Veins 5 ft. wide	1.60	3.00	
(b) Veins 10 ft. wide	1.25	2.50	
(c) Veins 25 ft. wide	1.00	1.75	
(d) Large deposits	.....	0.29–0.80	
Average exclusive of (d)			2.25
5. Cut-and-fill			
(a) Favorable conditions	1.25	2.50	
(b) Unfavorable conditions	2.25	4.25	
Average			3.40
6. Square-set			
(a) Favorable conditions	1.75–2.50	3.25–4.25	
(b) Unfavorable conditions	2.50–5.00	4.25–9.00	
Average			

Concreting the Denn Shaft

F. P. Brunel

■

Unemployment relieved and improvements of permanent value effected by a construction project effectively consummated while productive operations awaited a resumption of normal economic conditions

■

PLANS FOR CONCRETING the Denn shaft of the Shattuck Denn Mining Corporation, Bisbee, Ariz., were started in the early part of July, 1931, and pouring of concrete began on Sept. 4 of the same year. As the upper part of the shaft had been sunk about 30 years ago, the timbers were in such condition that the shaft had to be retimbered or concreted, if it was to be kept in condition as an operating shaft. Although concreting was by far the more costly, the management considered it to be the better plan. The shutdown caused by the low price of copper afforded an opportunity to do the work while no other activity existed at the mine, and enabled the company to keep a skeleton organization together, besides helping to relieve the unemployment situation in the district by maintaining a payroll of over $10,000 per month. Concreting was completed on June 18, 1932. Actual working time was 9 months and 14 days. Two shifts daily were used from Sept. 4 to 9, and thereafter three shifts daily. The work was done under the supervision of J. G. Flynn, general manager of the company. F. P. Brunel and J. A. Wilcox were the construction engineers, and A. Nicholson was shaft foreman. Costs are presented in the accompanying table.

The shaft is 2,242 ft. deep. From the collar to a depth of 1,350 ft. it is a three-compartment shaft, with a small pipe compartment. Dimensions of the hoisting compartments are 4 ft. 8 in. x 6 ft. in the clear. From the 1,350 level to the sump the shaft has four full compartments, but from the surface to the 1,350 level the dinky and pipe compartments are combined into one large compartment, measuring 6 ft. x 7 ft. 3 in. in the clear, with a channel iron grouted in every 5 ft. to provide support for the guides. Pockets were left in the concrete, and the channel irons were set in place after the forms had been pulled.

When the shaft was begun, the geological conditions of the district were not well known. Although sinking started in the lime area, at a depth of 1,800 ft. nevertheless the contact of the lime and schist was struck. From the 1,800 to the 2,000 level the shaft is directly in the junction of the Dividend and Mexican Canyon faults. The Dividend fault is the principal geological feature of the district, the fault having an estimated throw of 1,500 ft., and moving the Pinal schist on the north against the Paleozoic sediments to the south. The Mexican Canyon fault is one of the major faults of the district. At this point of the shaft the ground was heavy and difficult to hold, and the large amount of loose ground encountered necessitated driving of spiling and back walling. Old 4-in. pipe was used as stulls, as well as timber. The iron pipes were placed by inserting the lower ends in the concrete while pouring. They were allowed to lean in toward the walls of the shaft, with the tops inserted under the wall plates. The stulls prevented the timbers from slipping, and by placing lagging back of them, a good back wall was built, which not only prevented the flow of loose muck but also cut down the amount of concrete to be placed. The loose muck that fell on top of the bulkhead was shoveled into cars and hoisted to the surface.

In the first 1,000 ft. of the shaft the timber had been in place for a number of years, most of the blocking had rotted, and, owing to the lack of bearers, the timbers had to be supported at all times from below. Temporary bearers were placed, and the timbers reblocked. Hanger bolts were placed every set, for ten or twenty sets above where the men were working, before proceeding with the pouring of concrete. Standard hanger bolts were used where possible, but on account of numerous doubling-up sets in many places, bolts could not be used, so old 2-in. iron pipes, flattened at the ends and secured to the timbers with lag screws, were employed. Where needed, stulls that extended to the timbers above were placed on top of the last set poured, so as to prevent the timbers from slipping.

A total of 2,230 ft. of shaft was concreted. Maximum thickness of the walls was 3 ft., and the minimum, 10 in. Curtain walls were 8 in. thick, and, where the ground was heavy, they were reinforced with I-beams. Concrete placed totaled 9,934 cu.yd., which required 9,052 tons of crushed rock, 5,991 tons of sand, 10,934 bbl. of cement, 55,750 lb. of Celite.

Use of Celite facilitated the removal of forms, and thereby lessened the wear and tear on them. It also kept the aggregate in suspension, which assisted in the pouring. The size of each batch was ⅓ cu.yd., and the mixture used was: 1 part cement, 3 parts sand, and 5 parts rock. Each batch of concrete contained 1.51 cu.ft. of cement, 4.59 cu.ft. of sand, and 7.56 cu.ft. of rock. The average amount of concrete poured per month was 234.5 lin.ft. of shaft. Maximum footage for a 30-day period was 455 lin.ft., and the minimum was 115 ft. The low figure was the result of encountering heavy ground. Amount of concrete per shaft-foot averaged 4.47 cu.yd. A chart was kept showing the position of guide pockets, single and double, and the position of pockets for the I-beams that supported pipe and cable hangers. On this chart the number of batches used for each 5 ft. was recorded, also the date, the time, and the shift boss who did the pouring.

The crew on each shift consisted of seven men and a shift boss. While pouring, three of the crew came to the surface and operated the mixer and charging chutes. When pulling timber and forms, the entire crew worked in the shaft. An unusual situation in connection with the job was that no member of the crew quit or was discharged; the same crew that started the work completed it. Because of the unemployment situation in the district, a staggered plan of employment was used with the shaft and surface crews. The job was completed without any serious accidents, and with only a few lost-time accidents, due to injuries to hands and fingers.

After a set had been poured and the top lagged over tightly, the timbers were removed, the ground was made safe, and the forms were pulled. The forms were so designed that they could be dismantled easily. After being loosened, the lowest of the three sets used was hoisted into place on top of the last set poured. Extra forms were available on the surface to replace any that might be damaged. The forms were hoisted by fastening them to a chain suspended under the cage, special devices affording protection against their breaking.

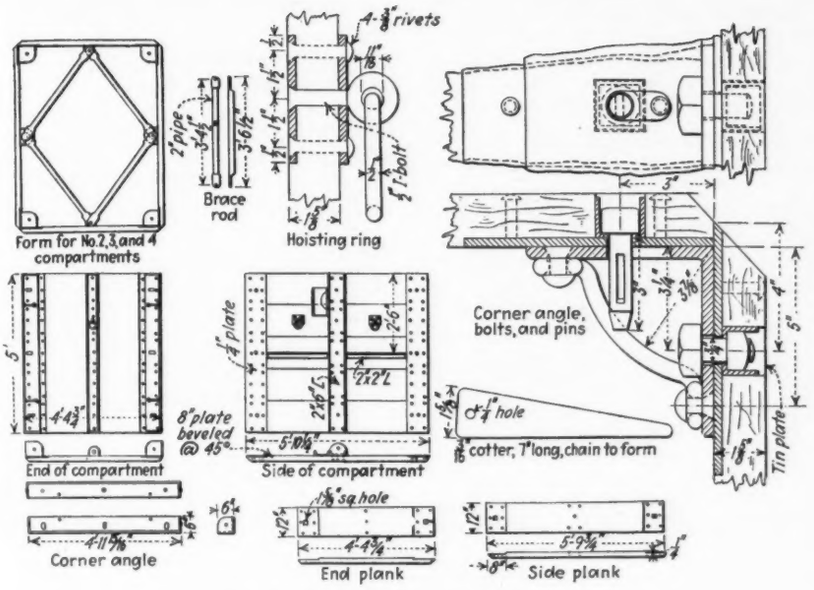


Fig. 1—General details of shaft forms

Two bulkheads were always used directly underneath where the men were working. These consisted of two 6-in. channel irons, inserted in pockets left in the concrete, and 2-in. lagging placed on top of the channel irons. The men, when they were engaged in fastening forms to the chain or removing channel irons and staging, were lowered in a sling suspended underneath the cage. In these operations they were required to wear life belts. The sling was of special design and similar to that used by boatswains. After the forms had been pulled and set in place, they were securely braced. Guide pockets and staging pockets were fastened to the forms, so that they would be in the proper position when the forms had been lined and squared. The type of staging-pocket and guide-pocket molds is described fully in another part of this article. The staging pockets were placed every 5 ft., two at each end near the walls of the shaft.

Wooden forms, reinforced with angle irons and channel irons, were constructed in the shops at the mine from 2x12-in. plank, planed and sized. The channel irons were riveted on, and the angle irons were bolted on. Corner irons were riveted on, so as to prevent breakage when prying them loose. Details of the forms are shown in Figs. 1 and 2. Before placing the first set of forms, a templet set was placed about 30 ft. below the 2,200 level. Ordinary shaft sets were used. Holes were drilled into the walls of the shaft, and long pieces of steel were inserted in the holes. The templet set was then placed on top of the steel, so as to have a good anchorage. Wooden cleats, 4x4 in., were fastened with lag screws to the face of the timbers, flush with the top. Then the forms were placed on top of the templet set, but resting on the cleats. Placing of the cleats facilitated the removal of the forms. Four sets of forms were constructed, but only three sets were used in the shaft. The extra set was used for replacement.

All stations were standardized, the forms being built in the shop and lowered to the stations when needed. Forms were so constructed that they could be set in place, and one section interlocked with another; no nailing was required. They were easily dismantled, and, when a station had been poured and the concrete had set sufficiently, they were removed and hoisted to the next station. An assembly of station forms is shown in Fig. 3. Station columns, brows, and curtain walls were reinforced.

The guide pockets were made of old 8-in. extra-heavy pipe, cut into 8-in. lengths with an acetylene torch. A semicircular plate of ½-in. iron was welded on the ends, leaving an opening for inserting a wrench or the hand. The guide pockets were held in place with a metal wedge that fitted into the opening of the pipe. This wedge extended out through the semicircular notch in the wooden form, and another wedge driven into the hole held the pocket firmly in place. A bolt was then inserted through the guide pocket and both the forms at the curtain wall, and then bolted tightly. This held the guide pocket rigid. On the end walls, the back of the guide pocket had a ⅛-in. plate welded on, to prevent entrance of concrete. A large amount of old pipe on hand was a factor in choosing this type

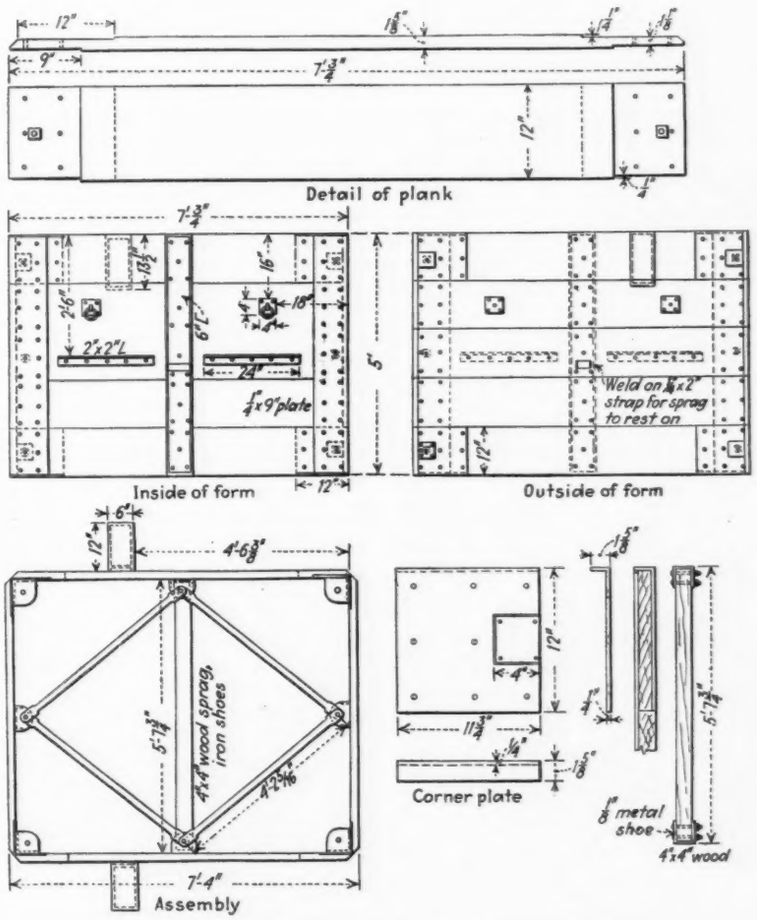


Fig. 2—Forms for combined dinky and pipe compartments

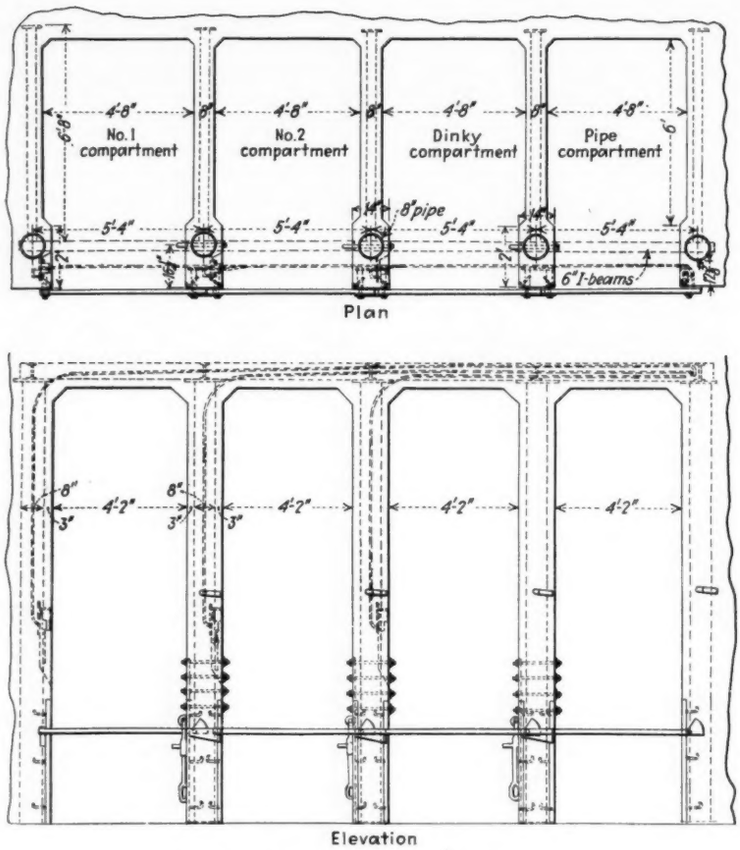


Fig. 3—Assembly of station forms

Summary of Shaft-Concreting Costs

Mixing plant		$1,585.95	
Material handling plant		2,140.98	
Concrete forms		10,824.12	
Concrete pouring column, 4-in. pipe		1,882.10	
Cost of preparatory work			$16,433.15
Direct labor		46,158.81	
Direct material:			
Cement, 10,934 bbl.	$33,569.37		
Sand, 5,991 tons	6,156.94		
Rock, 9,052 tons	11,864.61		
Celite, 55,750 lb.	793.02		
Steel	1,807.22		
Timber	2,495.19		
Miscellaneous	522.07	57,208.42	
Engineering and surface supervision		4,495.56	
Hoisting		19,249.57	
Testing materials		2,057.32	
Water and light		3,373.40	
Maintenance mixing plant		1,312.37	
Handling supplies		11,336.80	
Shaft guide pockets		3,988.64	
Miscellaneous		13,357.76	
Concreting cost			162,538.65
Total cost			$178,971.80

During the period of shaft concreting, no other activity existed at the mine, and, consequently, about $30,000 of expense is included in the above costs that would have been incurred by the maintenance of plant and organization, even though all operations had been suspended.

No salvage deductions were made in the above tabulation; however, considerable timber and steel used in constructing the surface plant was salvaged. The item showing the cost of forms includes maintenance cost. Deducting $30,000, the actual cost of concreting the shaft was $148,971.80.

of guide fastener. The fastener is simple, and seems to meet all requirements.

The ore storage bin of the old Shattuck mill served for storing rock and sand, which were unloaded directly from side-dump railroad cars. Each of the two compartments of the bin holds about six carloads of material. Conveyors in the mill were used to move the material to a small two-compartment bin, constructed underneath the conveyor. The conveyors were put in operation with a few minor changes of the old mill equipment. The cost involved was negligible.

After being loaded into mine cars, the rock and sand was transported to the shaft with a Goodman battery haulage motor. Sixteen cars per trip were hauled, and four trips could be made to the shaft in an hour. On reaching the shaft, the mine cars were hoisted to the top of the ore bins. An auxiliary headframe was constructed adjoining the steel headframe, and the capacity of the ore bins was increased. An overhead track was built so that the material could be dumped directly from the mine cars into the bins. Crushed rock went into one compartment, and sand into another. The cement was emptied into a large cylinder made out of a smokestack, the cylinder having a capacity of 12 tons of cement. Cost of handling material was low. By hoisting the material independently of the shaft, the hazard of material falling down the shaft was avoided, and the shaft cages were always available for the hoisting of shaft timbers and the pulling of forms.

Chutes for discharging material into the mixer were constructed underneath

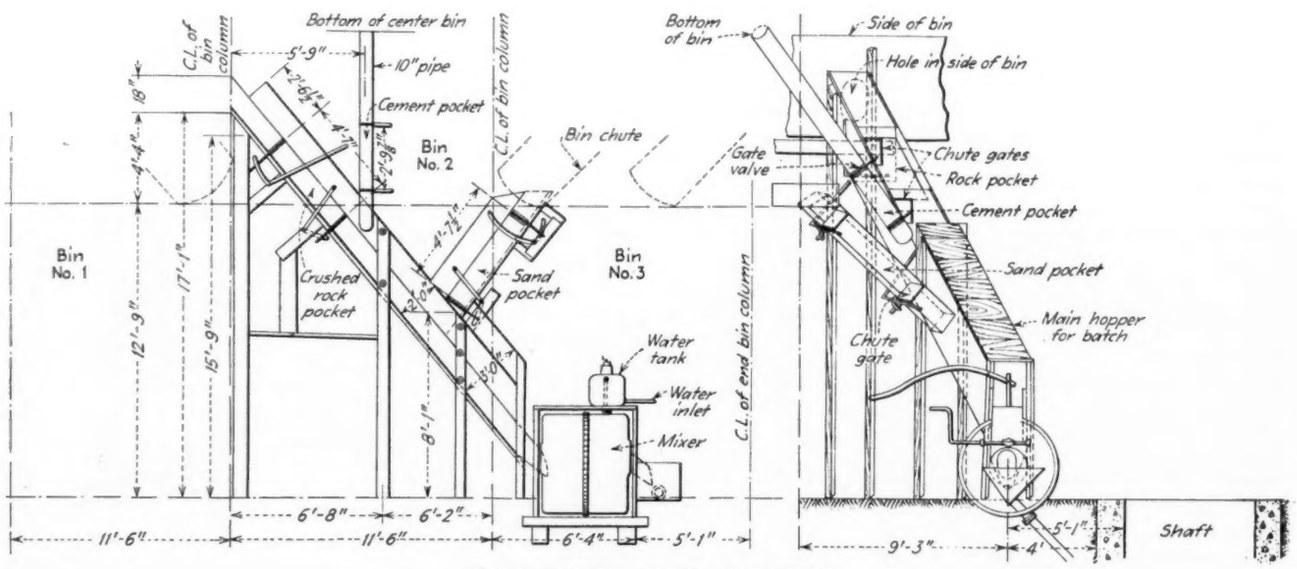


Fig. 4—Arrangement of surface plant

the bins. A large main chute, built for the crushed rock, connected the rock bin with the mixer. Chutes from the sand and cement bins emptied into the rock chute. Charging pockets that held the proper amount of materials for each batch were constructed in the chutes; they were filled and emptied automatically by the movements of gates. This method of charging insured a uniform mix at all times. No shoveling whatever was necessary. Valves for loading and unloading the charge were installed in a 10-in. pipe that connected the cement bin to the main hopper. The aggregate emptied into the main hopper, and, by opening a gate, was discharged directly into the mixer, when needed. From the mixer, the concrete was poured into a funnel, to which a 4-in. pipe was attached. The concrete flowed down the shaft through the pipe, and thence into a sinking bucket, equipped with a discharge spout. Iron launders conveyed the concrete into place. Fig. 4 shows the charging arrangement that was devised.

All timber in good condition was salvaged and stacked in the yard, the rest being given to the unemployed of the camp and to the employees of the company. The shaft contained an unusual amount of timber. It was cribbed solid for about 200 ft., and contained a number of doubling-up sets in the upper section.

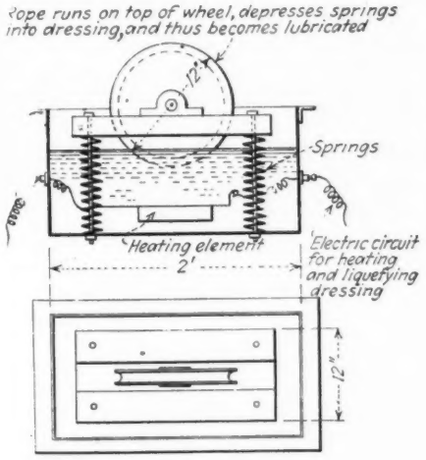
The go-devil for the placing of guides was built long enough to accommodate a 20-ft. guide. A steel ladder was fastened on the side, and four stagings were placed in it. The crew consisted of four men per shift. After spotting the holes with a plumb line, they were burned with an acetylene torch. In no instance were the guide pockets found to be more than ½-in. out of vertical. The lengths of guides were fairly uniform, as just a slight creep occurred.

■

Lubrication of Endless-Rope Haulage Lines

■

CONSIDERABLE DIVERSITY of opinion exists in mining circles on the Witwatersrand as to what degree and variety of lubrication of haulage rope will give the longest life under heavy working conditions. From tests made in a mine where 32,000 ft. of ⅞-in. diameter steel rope is in use, an increase in the life of the rope, better manipulation of the cars or trucks, and increased tonnage have been effected by the even distribution of lubricant.



Haulages of the endless-rope type were situated at deep levels in the drifts where humidity and temperature of the air were above normal, with water and mud fairly frequent. The trucks were side-tipping and fitted with ordinary gooseneck jockeys, deflecting and tension sheaves accommodating the rope at hook-on and hook-off points along the tracks. In the first test, extending over many months, with the rope entirely free from lubricant, about 8,000 ft. of new rope per month had to be spliced into the 32,000 ft. Numerous broken wires occurred because of distortion of the rope at the gooseneck, caused by failure to free itself.

The heart of the rope deteriorated rapidly, particularly at splices, because of moisture. Segments of the surge wheels on the haulage had to be replaced every three or four weeks, and numerous truck derailments took place, delaying output.

Since the introduction of lubricant these defects have almost been eliminated. Only about 1,000 ft. of new rope per month is spliced into the total length. Few broken wires occur; the heart of the rope is good. No surge-wheel segments have been replaced, and fewer derailments have taken place. A comparison of rope performance under the two sets of circumstances is shown in the accompanying table.

A simple drip-feed lubricator filled with heavy oil is used at the lead-in rope to the haulage when starting up after the week-end stoppage. As soon as the gears are in stride, the rope-dressing type of lubricator is used as shown in the sketch. Discarded oils from hoists, heavy-oil engines or petrol engines form the basis of the dressing, which is obtained at low cost, thereby insuring economical maintenance.

Comparison of Rope Performance

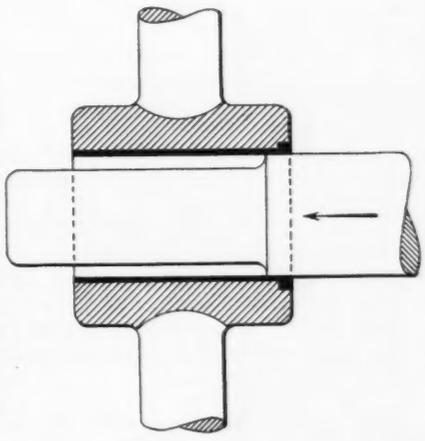
Unlubricated Rope	*Lubricated Rope*
New condition: ⅞-in. dia. 6/7 construction	New condition: ⅞-in dia., 6/7 construction
4 months' life	14 months life
4 weeks' replacement of surge segments	No replacements
Numerous track derailments	No derailments

■

Flanged Bushing Solves Problem

A FREQUENT maintenance job is tightening a wheel on its shaft. In most instances this can be done by truing the wheel, boring the hub, and fitting a bushing. To get the wheel and bushing in place on the shaft, and to secure a tight fit, is not so simple, especially if one tries to save the keyway in the wheel and shaft and use a thin wall bushing.

At one Western mine shop the master mechanic turns the bushing with a small flange on one end, about ⅛ in. high and ¼ in. long. When the bushing has to be flush with the face of the hub, the latter can be bored to take the flange. When assembling, the flange is on the side from which the shaft is pressed.



Developments in Flotation-Cyanidation

L. E. Tucker and S. J. Swainson

FLOTATION of base-metal sulphides had been practiced for years before the process was applied to precious-metal ores, to the recovery of sulphides containing precious metals, and to the removal of cyanicides prior to cyanidation. The successful application of flotation to the treatment of metallic copper ores indicated that free gold might be similarly recovered. The fact that native copper could be floated led to analogous reasoning in respect to native gold. Refinements in flotation practice originally developed in the selective flotation of base-metal sulphides and metallics are now available for precious-metal flotation. Outstanding among these refinements has been the development of modern chemicals to replace the old oil mixtures. As a result, the flotation of native gold is an established method of recovery. Ratios of concentration in modern gold flotation are generally higher than in base-metal flotation; in fact, ratios of 100-150 to 1 are not uncommon. Recoveries approximate the results of cyanide practice.

Precious-metal ores generally have a much wider variation in valuable components than do base-metal ores, and the physical characteristics of the valuable components are extremely important. The metallurgist's problem is to adopt those processes that will give the greatest dollar yield per ton of ore treated.

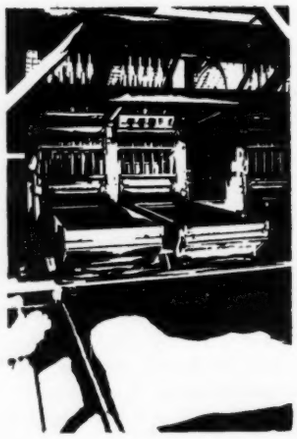
From the standpoint of metallurgical treatment, the precious-metal ores may be classified as follows:

1. Ores containing free or native gold in quartz or quartz-adularia gangues. These ores seldom contain appreciable amounts of silver.
2. Ores in which the gold and silver are associated with the oxides of lead, copper, and iron.
3. Ores in which the gold and silver are associated with pyrite or other sulphides. The majority of precious-metal ores fall in this class.
4. Ores that contain gold and silver with appreciable quantities of arsenic, antimony, or tellurium.
5. Ores in which the gold and silver are associated with the common sulphide minerals of copper, lead, and zinc, the precious metals being for the most part of secondary importance in the consideration of the most economical method of treatment.
6. Tailing, current or old.

In applying concentration to the treatment of any of these classes of precious-metal ores, several processes and combinations of processes may be applicable from the metallurgical standpoint, but the economic status of a given operation usually limits the choice. A mine is a depleting asset, so a maximum dollar yield must be secured with minimum capital outlay. The most common groupings of treatment methods for precious-metal ores are the following: (1) Amalgamation and flotation; (2) amalgamation and cyanidation; (3) sand leaching and cyanide; (4) all-slime cyanidation; (5) gravity concentration and cyanidation; (6) gravity concentration and flotation; (7) flotation and cyanidation of tailing; (8) cyanidation and flotation of tailing; (9) flotation followed by cyanidation of concentrate; (10) straight flotation.

Table I—Flotation of Amalgamation Tailing

	1	2	3	4
Tons per day...	260	210	100	60
Heads, Au.....	$1.60	$1.80	$4.50	$3.00
Concts., Au.....	$100	$120	$275	$275
Tailing, Au.....	$0.28	$0.28	$0.50	$1.40
Recovery, per cent.....	82.70	84.60	89.00	53.60
Ratio of concentration....	75 to 1	79 to 1	68 to 1	171 to 1
Grinding, per cent + 100 mesh.....	3.0	15.0	3.0	30.0
Reagents lb./ton:				
Aerofloat 15.....	0.12	.....	.....	0.14
Aerofloat 208....	0.005	0.020	.....	.....
Reagent 301....	0.002	0.004	0.23	.....
Amyl xanthate..	.....	.....	.....	0.01
Cresylic acid....	.....	0.25	0.16	.....
Copper sulphate.	.....	0.04	0.09	0.07
Soluble starch...	.....	.....	0.80	0.80



Amalgamation and flotation has usually been built up in the older mills that at one time practiced amalgamation and gravity concentration. By replacing gravity concentration of the amalgamation tailing with flotation, additional recoveries in a higher-grade concentrate have been realized, and the net result has been profitable, the flotation equipment installation being paid for from a few months' operating profit. Table I shows examples of results obtained by the flotation of amalgamation tailing.

Amalgamation followed by cyanidation is probably the most efficient of all combinations from the metallurgical standpoint. Plant installation costs are high, and most of the operating plants are old mills whose practice has been built up from experience rather than from testing of known orebodies. In general, these plants employ stamps or ball mills for secondary crushing, followed by stationary plates for catching coarse gold. Tailing from the amalgamation plant is classified into sand and slime, and the sand leached and the slime agitated for recovery of the remaining gold by cyanide solution. Coarse free gold is trapped in the battery and on the amalgamation plates as soon as liberated. The fact that the entire production of this type of plant is bullion is distinctly advantageous for isolated operations. Choice between flotation and cyanidation depends on such factors as actual recoveries, geographical location of the plant, cost of plant, and life of orebody.

Gravity concentration and flotation is applicable to the same type of ores as gravity concentration and cyanidation. The choice between cyanidation and flotation depends on such factors as recovery, geographic location of plant, and cost of plant. Flotation-cyanidation is particularly applicable to ores which contain cyanicides or refractory minerals that do not yield to straight cyanidation. In some cases flotation precedes cyanidation and recovers into a small bulk the cyanicides or refractory constituents to which special treatment can be given. The flotation tailing, containing the more amenable portion of the precious metals, is then cyanided. Two mills in Colorado have successfullv applied this scheme to the treatment of ores containing tellurides. In one case the tellurides are floated in a weak cyanide solution and a high ratio

Abstract of Technical Paper No. 20, American Cyanamid Company.

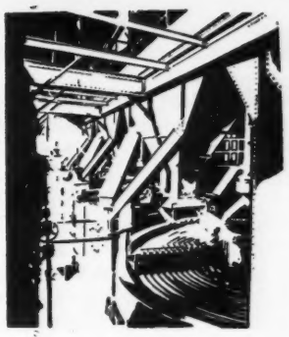
of concentration is obtained. This concentrate is roasted prior to cyanidation, whereas the tailing passes directly to the cyanide circuit. In another mill the tellurides and sulphides are floated in water, and about 75 per cent of the gold is recovered, with a concentration ratio of 20-35 to 1. Concentrate is roasted prior to cyanidation; the tailing is dewatered and cyanided with the roasted concentrate. Obviously this scheme of treatment is much cheaper than roasting the entire ore.

A large operation in Central Canada is treating a copper-zinc-gold ore by flotation, followed by cyanidation of the flotation tailing. Copper and zinc are selectively floated, pyrite being depressed, after which the tailing from the zinc circuit is dewatered and cyanided for the recovery of the gold associated with the pyrite. A small operation in the Eastern Transvaal is treating a cupriferous gold ore by the same scheme. The sulphide copper minerals are floated, after which the tailing is cyanided. An operation in Guerrero, Mexico, is treating a silver ore, containing base-metal sulphides, by flotation followed by cyanidation. The flotation tailing is classified into slime and sand, the slime treated by agitation and the sand leached by percolation.

One of the largest gold properties in Eastern Canada, which formerly employed straight all-slime cyanidation, has recently adopted the combination cyanidation-flotation scheme of treatment. Flotation in this mill follows cyanidation and recovers the refractory portion of the gold not recovered in the cyanide circuit. The introduction of flotation at this property has reduced the cyanide tailing by more than 50 per cent. The flotation practice is as follows:

The cake from the last stage of filtration in the cyanide plant is repulped with fresh water and pumped to a conditioning agitator, where the pulp density is reduced to 33-35 per cent solids. Sufficient sulphuric acid is then added to bring the pH to 7.4-7.7. Promoter is then added in the amount of 0.15 lb. per ton. The pulp is then pumped to a Pachuca vat, where it is given a further period of conditioning, the total period being 45 min. From the Pachuca the pulp passes through a distributor and thence to eight 10-cell mechanical rougher machines, where sufficient pine oil is added to give the desired froth volume. The rougher concentrate is cleaned in one 8-cell mechanical machine. Concentration ratio is around 100 to 1, giving a concentrate assaying about $250 gold. The final tailing assays 55c.

The practice in the concentrate-cyanidation plant is a new development in the treatment of tellurides. No roasting is done. Instead, the concentrate, after dewatering, is ground in a 5x16-ft. tube mill lined with silex and charged with pebbles. The mill is in



closed circuit with a 10-ft. classifier which is similarly in closed circuit with a 24-ft. Dorr hydro-separator. The overflow from the hydro-separator is extremely fine, about 60 per cent being finer than 10 microns. This product is subjected to four stages of agitation in turbo agitators, each stage of agitation being followed by thickening. Overflow from the thickeners is pumped to the main cyanide-mill solution storage. Discharge from the last thickener is final tailing, sent to waste. Aero brand cyanide is fed to the grinding circuit in sufficient amount to maintain a solution strength of 0.075 per cent KCN equivalent. Lime is added at various points in the circuit in sufficient amount to maintain the solution nearly saturated. In spite of the fact that this concentrate contains a high percentage of gold tellurides, the extraction is about 98 per cent.

Flotation of the original ore, followed by cyanidation of the concentrate, is an economical process for ores that yield a final tailing by flotation. To predict, without actual experimentation, whether or not a given ore will yield a satisfactory tailing by flotation is impossible. In general, those ores that contain precious metals as sulphides, sulpho-arsenides or antimonides, as tellurides, or in intimate association with sulphides such as pyrite or arsenopyrite, are amenable to flotation. Where this method of treatment gives satisfactory recovery, it is cheaper than straight all-slime cyanidation. For example, when a large gold property in Eastern Canada increased its mill capacity, flotation, followed by cyanidation of the concentrate, was adopted in place of straight cyanidation. The cost of the flotation-cyanidation plant was only two-thirds of the cost of a cyanide plant having the same capacity. In addition, the milling cost has been reduced by 20 per cent. The following is a brief description of the practice at this plant:

The ore is a sericitic schist containing about 7 per cent sulphides, mostly pyrite. The gold content varies from $7 to $9, part of which occurs as free gold, the remainder being associated with the pyrite. The jaw-crusher product from the underground crushing plant is passed through the secondary crushing plant, which consists of one 7-ft. Symons cone and a set of 78x18-in. rolls, in closed circuit with 4-mesh vibrating screens. The undersize from the 4-mesh screens forms the feed to the fine-grinding section, which consists of five 5x16-ft. tube mills, each of which is in closed circuit with a 6x24-ft. Dorr duplex classifier. Promoter, to the extent of 0.12 to 0.15 lb. per ton, is added to the grinding circuit.

During the first month of operation the free gold had a marked tendency to build up in the grinding circuit. The classifier sand, for example, was found to contain from six to eight times as much gold as the original feed. Experiments showed that a considerable portion of this gold could be recovered by flotation of the tube-mill discharge. Accordingly, one mechanically agitated flotation cell was placed at the discharge end of each tube mill. A portion of the tube-mill discharge, diluted to about 30 per cent solids, is treated in this cell. A small amount of Aerofloat 25 is added to the pulp as it enters the cell. Enough gold is recovered in this single cell to maintain the classifier sand at about the same value as the feed. Microscopic examination of the concentrate shows that a considerable amount of plus 35-mesh free gold is being recovered. The concentrate is dewatered with the main flotation concentrate, and the tailing passed to the classifier. Although this additional flotation operation has not reduced the final tailing appreciably, the tailing has been more uniform in gold content.

The classifier overflow, maintained at about 30 per cent solids and averaging 12 per cent plus 65 mesh and 55 per cent minus 200 mesh, passes to the primary flotation circuit. This consists of 48 mechanical cells, which produce a final concentrate assaying $55 to $60. The tailing passes to a secondary flotation circuit having the same number of cells. This secondary concentrate, which assays $14, is returned to the head of the primary circuit. The tailing assays $0.29 to $0.30 and is sent to waste. A total of 0.12 lb. Aerofloat 25 is added at various points in the flotation circuit.

The method of thickening the concentrate is unique. The apparatus consists of one 12x12 Dorr agitator, from which the air lift has been removed. The dewatering is effected with twelve 5x3.5-ft. Butters filter leaves, submerged in the pulp. Vacuum of 24 in. is supplied by the same pump that serves the filters. A 1½-in. cake builds up on the filter leaves. This is removed once in 24 hr. by raising the leaves and scraping them. The concentrate enters the thickening apparatus at 33 to 35 per cent solids and has the following percentage screen analysis: plus 48 mesh, 0.2; plus 65 mesh, 4.2; plus 100 mesh, 11.8; plus 150 mesh, 12.8; plus 200 mesh, 17.8; minus 200 mesh, 53.2. The discharge, containing about 60 per cent solids, is drawn from the bottom of the agitator by a centrifugal pump and is sent to two 8-leaf American filters,

where the cake is sprayed with water to displace as much as possible of the frothing reagent prior to cyaniding.

The filter cake, containing 20 per cent moisture, is repulped with barren solution and ground in two 5x16-ft. tube mills lined with rubber and charged with 1¼-in. balls. Each is in closed circuit with a 6-ft. Dorr duplex classifier, the overflow of which, having a pulp density of 50 per cent solids, goes to five 24x20-ft. Dorr agitators in series. Each agitator is given auxiliary aeration in addition to that supplied by the standard air lift. Time of treatment in the agitators is about 48 hr. The discharge from the last agitator goes to a bowl classifier, the sands from which are returned to the regrind mills. The bowl overflow, which is all minus 200 mesh and 95 per cent minus 325 mesh, goes to one 24x20-ft. Dorr agitator and thence to a 50-ft. Dorr tray thickener. The upper tray is used for collecting the froth that builds up. Pregnant solution, assaying $10 to $11, is drawn off the bottom tray, sent to clarification, and thence to Merrill precipitation presses. Barren solution assays 4c. per ton. Thickener discharge is further dewatered in six 8-leaf American filters operating in three stages of two each. Final tailing from the cyanidation of the concentrate assays $0.85 to $1, and the assay value of the combined flotation and cyanide tailing is 39 to 40c. The solution in the cyanide circuit is maintained at 0.20 per cent KCN equivalent, and 0.075 to 0.10 per cent CaO.

A property in Western Australia, which formerly treated telluride ores by roasting and cyanidation, has abandoned this method in favor of flotation followed by roasting and cyanidation of the concentrate. As a result of this change costs have been materially reduced and the tonnage of profitable ore has been correspondingly increased. The practice at the new flotation-cyanidation plant is as follows: The ore is crushed in a gyratory crusher and Symons cones to pass ½-in. screen. This product is further reduced in five 6x5-ft. ball mills in open circuit. The ball-mill discharge flows over corduroy strakes, where coarse free gold is removed. From the tables the pulp passes to four drag classifiers, each of which is in closed circuit with a 5x22-ft. tube mill. The classifier overflow, 85 per cent minus 200 mesh, is thickened and then passes through two 8-cell mechanical flotation machines, where tellurides and pyrite are recovered. The only reagents used are 0.09 lb. thiocarbanilide, added to the tube mills, and 0.05 lb. Aerofloat 25, added to the flotation cells. The ratio of concentration is 10 to 1, and the concentrate assays $75. The concentrate is filtered, dried, roasted in Edwards furnaces, and cyanided in the usual manner.

A plant in the Western United States is cyaniding 30 tons per day of pyritic concentrate assaying $58 gold. The concentrate is ground in two 4x9-ft. mills in closed circuit with a 6-ft. Dorr classifier that discharges 98 per cent minus 325 mesh overflow.

About 7 lb. lime and 9 lb. per ton of Aero brand cyanide are added dry to the tube mills. The classifier overflow discharges to one of six 13x10-ft. combination Devereux agitators and thickeners equipped for decantation. Treatment runs from 72 to 96 hours, with three to four decantations and washings. The final tailing assays $3.40. At another plant, 30 tons of concentrate, consisting of mixed table and flotation products, is ground in a 4x12-ft. tube mill in solution. This mill is in closed circuit with a table that makes a high-grade concentrate of galena, gold, and tellurides, averaging over $2,000 per ton. This is melted. The middlings are returned to the mill for regrind. Slime is sent to two 12x10-ft. Devereux agitators in series, then to three 14x10-ft. Dorr tanks, counter current. Heads vary from $50 to $100; tailing runs from $1.50 to $2.

The introduction of tables in closed circuits with the ball mill is not new, but it offers opportunity to eliminate the high-grade mineral that is free, and the more refractory minerals of high specific gravity. The use of tables to replace classifiers in the cyaniding of concentrate will depend on the refractory nature of the minerals in the concentrate. In some cases it is more economical to separate a portion of the concentrate into a high-grade product which can be shipped. If shipping is not feasible, all the minerals must be ground and the pulp cyanided with special treatment.

Straight Flotation, Concentrate Smelted. This method of treatment is finding wider application as the higher-grade ores are being depleted. Simplicity of design and low erection costs are deciding factors in adopting this method of treatment. It is particularly adapted to small operations with limited capital.

Table II—Straight Flotation, Heavy Sulphide Ores

	1	2
Tons per day	200	160
Heads, Au	$9	$18
Concentrates, Au	$83	$330
Tailing, Au	$0.65	$1
Recovery, per cent	93.50	94.70
Ratio of concentration	10 to 1	20.5 to 1
Per cent plus 100 mesh	9.0	13
Reagents lb./ton:		
Cresylic acid	0.11	0.12
Reagent 301	0.45	0.05

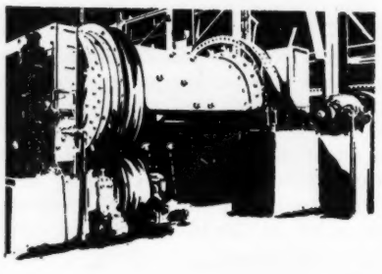


Table III—Straight Flotation, Precious-Metal Ores

	3	4	5
Tons per day	40	30	60
Heads, Au	$12	$7	$9
Concentrates, Au	$320	$125	$350
Tailing, Au	$1	$0.40	$1
Recovery, per cent	92.0	94.5	89.2
Ratio of concentration	29 to 1	19 to 1	44 to 1
Per cent plus 100 mesh	18.0	8.0	23.0
Reagents lb./ton:			
Aerofloat 15	0.13	0.10	0.06
Aerofloat 208	0.02	0.03	0.02
Reagent 301	0.07	0.04	0.02
Cresylic acid	.....	.....	0.06
Copper sulphate	.....	.....	0.05
Soluble starch	.....	.....	0.10

Flotation requires a variety of chemicals, divided into several groups according to their function:

Conditioners or Addition Agents: ammonium phosphate, copper sulphate, Aero brand cyanide, sodium cyanide, lime, soda ash, sodium silicate, sodium sulphide, sodium sulphite, zinc sulphate.

Promoters: Aerofloat 15, Aerofloat 25, Aerofloat 208, sodium Aerofloat, Barrett No. 4, Barrett No. 634, Reagent 301, Reagent 238, thiocarbanilide, T & T mixture, ethyl xanthate, amyl xanthate.

Frothers: Aerofloat 15 and 25, Barrett No. 4 and 634, cresylic acid, pine oil.

The most important factors, besides reagents, in precious-metal flotation are grind and density of the flotation feed. These must be determined for each individual ore. In general, gold flotation is not particularly effective on particles coarser than 60 mesh, and although feeds coarser than this can be readily handled, the greatest recovery is made on the finer sizes. With coarse feed, high pulp density should be used in order to prevent segregation of the coarse middling and heavy metallics during flotation.

In the field of flotation machines, present practice seems to indicate that the mechanical machines with high-speed impellers are in more general use than the air machines. This is probably due to the fact that, where metallics are present, they are not as stable in the froth system as the sulphides, and must be given every incentive to float. If the ore contains only a small amount of sulphides and a relatively large amount of precious-metal metallics, good practice is usually to carry a high water level and low froth column, so as to remove metallics as soon as they enter the froth system. Cleaning of precious-metal concentrate is likely to result in loss of gold, because, when the rougher concentrate is diluted, the metallics and coarse minerals tend to drop and enter the circulating load. For this reason a middling-return system is usually better than a roughing-cleaning system.

A survey of recent developments in the treatment of precious-metal ores shows that the trend is toward the use of a combination of flotation and cyanidation. The fact that many all-cyanidation operations have, during recent months, included flotation in their scheme of treatment is a strong argument for careful consideration of its possibilities by all operations now treating or contemplating the treatment of precious-metal ores.

Interpretation of Fault Movements From Mineral Fractures

Donald M. Fraser

FAULTING, with the resulting displacement of orebodies, is a common ailment of mining districts. Unfortunately, to determine the direction in which an orebody has moved along a fault plane is often difficult. The common criteria are drag along the fault, grooves, striæ, plucking, jointing, crescentic gouges, chatter marks, and gash veins. The megascopic evidence of the fault surface may furnish all the information desired. In

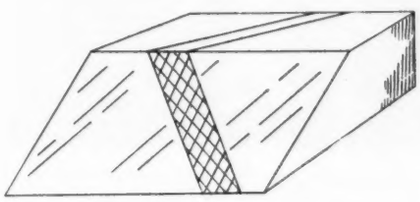


Fig. 1—Diagram of a faulted orebody.
Striae on the fault plane (the inclined front of the block) give the line of movement but not the relative direction of movement of the adjoining fault blocks.

some instances, though striæ give the line of movement, the determination of the relative movement of the adjoining fault surfaces is difficult. For example, in Fig. 1 the inclined front of the block represents the striated fault surface of a footwall block. The striæ show that the footwall, relative to the hanging wall, has moved either up to the right or down to the left. Of course, to explore with diamond drills in each direction is possible, but another method, much less expensive, is available. This is to determine, by means of microscopic examination of thin sections, the stress relations and the relative direction of movement of the adjacent sides of the fault.

Fractures in minerals, as observed with the microscope, are common features of crystalline rocks,[1] and are usually stated to be an indication that dynamic stress acted upon the rock some time during its history. While examining thin sections of gneiss and related rocks taken from zones of movement along the margin of a magnetite orebody and from fault zones in the same district, certain features were noted which led me to hope that fracture "sets" in the minerals might be used in the same manner as fractures in the rocks in interpreting stress relations associated with the faulting. The following conditions were observed in the thin sections:

1. Mineral grains are often broken along relatively smooth planes rather than along irregular surfaces.

2. In minerals having perfect cleavage, for example biotite, smooth fractures may intersect the cleavage.

3. Traces of the fracture planes, in some instances, occur in groups in which all are more or less parallel to a common direction, and thus form a set.

4. Often two sets of fractures occur, intersecting each other within a mineral grain.

5. Where only one set of fractures is present, it may closely resemble fracture cleavage.

6. Fracture sets in brittle minerals—that is, quartz—sometimes intersect approximately at right angles, but in general form obtuse-acute angles in less brittle minerals—that is, biotite.

7. Folds in some mineral grains—that is to say, biotite.

A biotite grain in specimen No. 6 (Figs. 2 and 3) illustrates several of the features described in the foregoing. It is approximately 2x6.5 mm. in size, and is bounded at one end and along part of each side by large fragments of what, before crushing, was a single crystal of albite. At the other end are several smaller grains of biotite, orthoclase, and magnetite. The rock slice as a whole has a massive structure, which is, of course, fractured. It was taken from a pegmatite dike cutting gneissic country rock, and was obtained along a fault zone which in places was marked by 18 in. of crushed material.

The end of the grain, which is intergrown with magnetite, orthoclase, and other biotite grains, shows the greater disturbance. Here it is crumpled and sheared. Two sets of shear planes intersect so that the obtuse angle between them is bisected by the trace of the cleavage. In places the shear planes are quite close together, and form a series of acute M's or W's. Near the middle and close to the opposite end of the grain is a distinct arching of the lamellæ to produce small folds. Occasionally fractures intersect the cleavage at, or close to, 90 deg. Most of these are the result of crumpling of a very short part of the lamellæ to produce microscopic monocline or terrace structures.

A small magnetite vein near the middle of the grain lies parallel to the cleavage. It appears at first glance to be fractured, but upon closer examination is found to be a composite vein, being made of several small lenses.

If the strain ellipsoid is applied to the features described in the foregoing, the following orientation is obvious. That the direction of easiest relief is in the plane of the section and perpendicular to the cleavage is shown by the small folds and the displacement along shear planes (see Fig. 3). The trace of the cleavage is the direction of maximum stress and least strain, and the perpendicular to the plane of the section is the mean stress axis. The stress was non-rotational and was applied parallel to the cleavage of the grain.

Considering Hartmann's Law, we find that the grain failed as a ductile or non-brittle material; that is, the direction of maximum stress bisects the obtuse angle between the planes of maximum shear or minimum distortion.

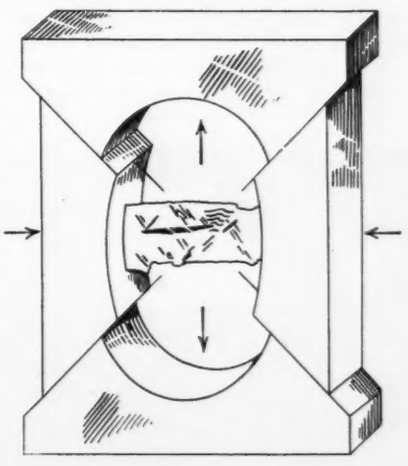


Fig. 3—Diagram of biotite grain reproduced in Fig. 2
This sketch shows the relative movement of the mosaicked fragments of the grain. Maximum stress was applied from the right and left, and easiest relief was to the top and bottom of the diagram.

Specimen No. 22 (Fig. 4) shows a biotite grain which is intensely sheared. The grain is ¾ mm. wide and 5 mm. long. Along one side is an irregular

[1]Bruno Sander discusses and interprets in great detail many mineral structures in his excellent book "Gefügekunde der Gesteine." Julius Springer, Wien, 1930.

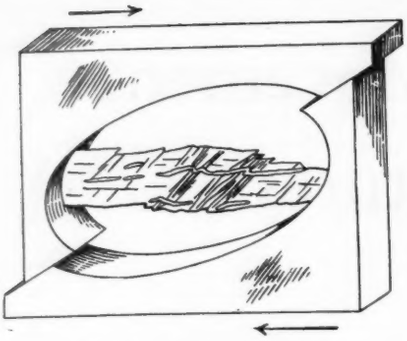


Fig. 4—Diagram of biotite grain

This sketch shows the relative movement of the ends of the grain. A rotational stress applied from the left at the top and from the right at the bottom caused the failure. The direction of greatest elongation, or the maximum strain axis, is the line bisecting the angle between the cleavage shear-plane and the fracture shear-plane.

mass of magnetite, at the ends are oligoclase and magnetite grains, and the other side is in contact with the balsam of the slide.

Source of the section is the crushed gneissic rock along the fault from which specimen No. 6 was taken. The hand specimen was massive, and showed streaks of biotite only locally. The grain under consideration is the largest in the section, and is more nearly parallel to, than normal to, other biotite grains approaching it in size. Cleavage planes are roughly normal to the plane of the section, and the acute bisectrix X lies in the horizontal plane. The obtuse bisectrix Z is also in this plane and Y is vertical to the section. The pleochroic formula is: X—yellow, Y—light cinnamon brown, Z—darker cinnamon brown.

The ends of the grain are more intensely sheared than is the middle part. At both ends the shear planes feather out to produce a horse-tail structure resembling, in miniature, the horse-tail faults of the Butte district. The average angle between the long axis (cleavage trace) and these close-spaced shear planes is less than 45 deg.

Near the middle of the crystal the fractures intersect the traces of the cleavage at angles of 35 to 60 deg. A few small tension fractures occur, intersecting those of the set already described.

Cutting the grain longitudinally is a small vein of feldspar, which is close to andesine in composition. It is folded into several anticlines and synclines, two of the former being definitely overturned so that the traces of their axial planes lie parallel to the trace of the shear planes in the section.

Applying the principle of the strain ellipsoid, difficulty is experienced in relating the definitely overturned folds, the single pronounced set of shear planes, and the varying direction of certain of the curved shear surfaces to the strain ellipsoid resulting from a simple non-rotational stress. If, however, a rotational stress is postulated, the features observed in the specimen are not only reasonable but expectable.

In the same section, a grain of albite-oligoclase (see Fig. 5), 2x2½ mm., is surrounded by grains of quartz, feldspar, and biotite, except where in contact with balsam at the edge of the slide. The albite-oligoclase, which shows polysynthetic twinning bands, is separated from the biotite described in the foregoing by a grain of magnetite and one of orthoclase, and is from 1 to 2 mm. away.

A conspicuous set of fractures or joints cuts the grain at an angle of 70 deg. to the twinning bands, which themselves mark a second direction of more poorly defined and less continuous fractures. The strain rhombs produced by these breaks indicate that the maximum stress was applied to the upper edge of the grain from left to right and along the lower edge from right to left. The relations of the strain ellipsoid to the fractures and axis of maximum elongation are shown.

Inasmuch as the last two grains—that is, the biotite and albite-oligoclase grains just discussed—are in the same section, any conclusions regarding the direction of application of maximum stress not in harmony would be absurd. Note that the best-developed fracture system of each grain dips in the same direction, but is more nearly vertical in albite-oligoclase than in biotite. This variation may be analogous to the difference in failure of shale and quartzite, where the former develops

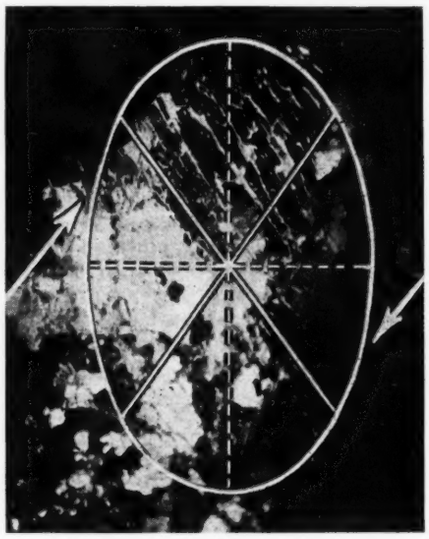


Fig. 5—Albite-oligoclase grain between crossed nicols. Mag. 14.

A well-developed set of fractures extends from the upper left to the lower right, and a second poorly developed set parallel to the twinning bands extends from the lower left to the upper right. The stress was rotational and roughly parallel to the less perfectly developed fractures. Movement on the left was upward and on the right downward.

Fig. 2—Faulted orebody. Mag. 20. Lighter patch in upper part of grain near the right-hand end has a different extinction position and marks a small fold in the laminae. Note diagonal fractures cutting the grain.

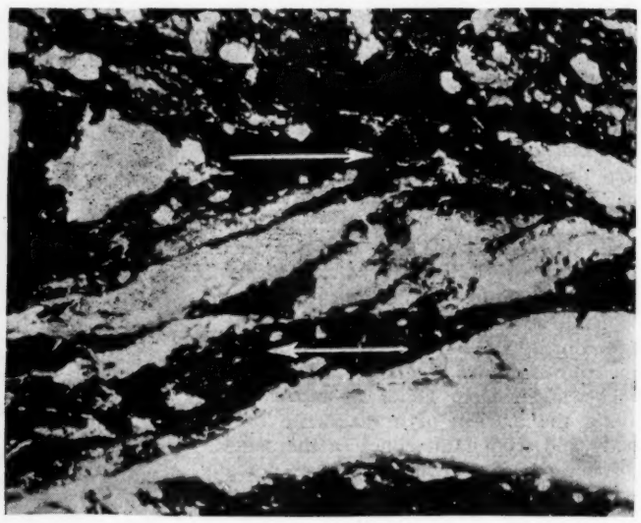


Fig. 6—Faulted orebody. Mag. 20. Lighter patches of quartz and feldspar elongated and tilted by movement, which these areas indicate to have been as shown by the arrows.

fracture cleavage and the latter develops joints having a higher angle of dip than the cleavages. If this relation is true, biotite may reasonably be classified as a relatively incompetent, and albite-oligoclase as a relatively competent, mineral.

in a direction corresponding to that which was assumed for the stresses producing or at least influencing the development of the gneissic structure of the specimen.

A number of observations such as those discussed in the foregoing led me to the belief that these mineral structures could be used in the interpretation of movements along faults. With this thought in mind, a number of specimens were collected from a fault exposed in the underground workings of the Alan Wood Mining Company's magnetite mine near Dover, N. J. This fault was chosen because determination of the relative movement of the walls from megascopic evidence was possible. The specimens were carefully marked, so as to preserve their orientation in relation to the plane of movement. Also, considerable care was exercised in obtaining sections of known orientation in relation to the fault.

Six sections were cut from four different specimens. These were studied, and the stress directions determined before referring to the markings which would indicate the orientation. The results were as follows:

Two sections showed no suitable structures. Four sections showed structures from which the maximum stress directions were determined to be the same for the four specimens and in agreement with the previously determined stress directions along the fault. Some of the structures observed in these specimens are described in succeeding paragraphs.

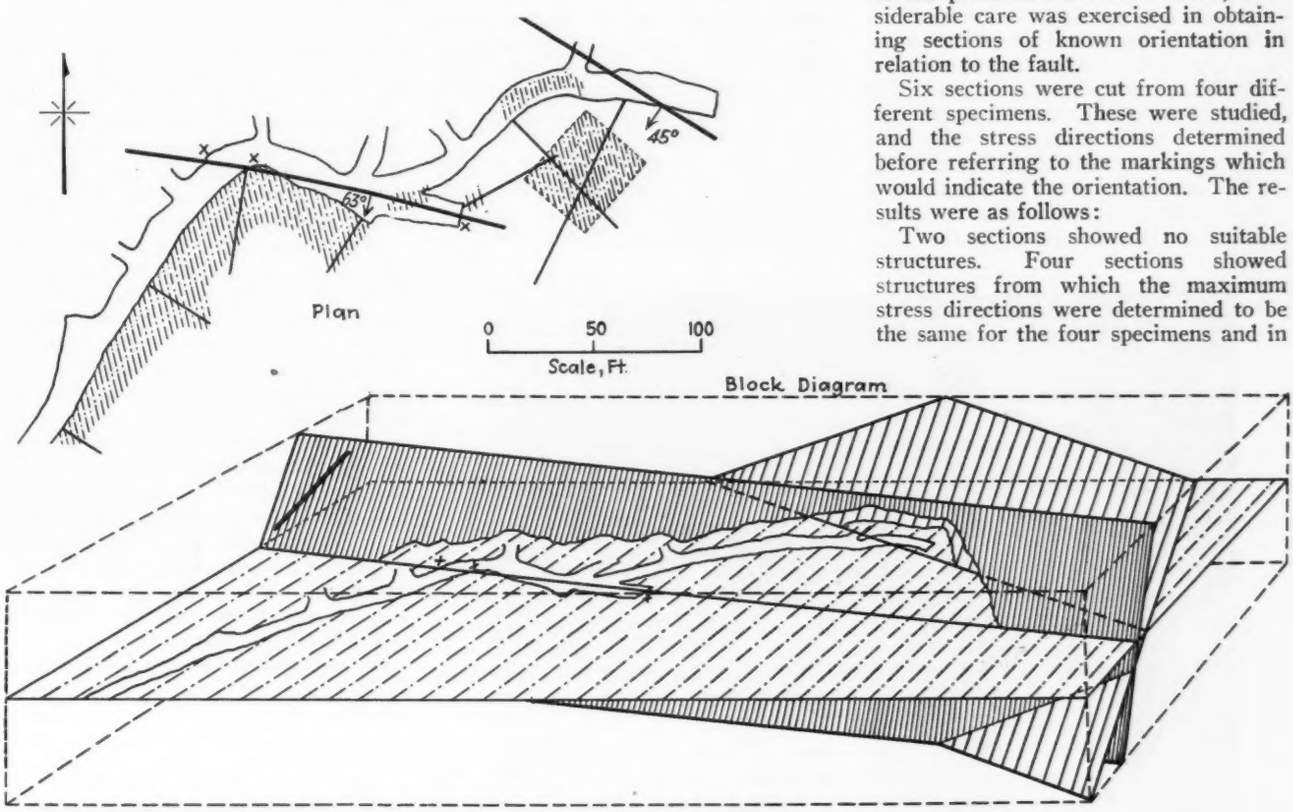


Fig. 8—Plan and block diagram of part of No. 2 level and two faults that intersect it. Specimens discussed in the text were taken from the fault of greater dip. Heavy line on the fault plane at the left marks the attitude of the striations on the fault surfaces. Fault planes have been cut out in part to expose the rear part of No. 2 level.

A section of a biotite grain in slide F-20 is approximately $1\frac{3}{4}$x$\frac{3}{4}$ mm. in size, and is oriented with the acute bisectrix X normal to the plane of the section Y parallel to the short axis of the grain and Z parallel to the long axis. The pleochroic formula is: X—yellow, Y—cinnamon brown, and Z—slightly darker than Y. It lies in the midst of quartz, feldspar, and other biotite grains. All of the latter are oriented with their long axes parallel to each other, and at 90 deg. to the long axis of the biotite grain under consideration. The rock is a gneiss, and the streaking due to the oriented grains is apparently at right angles to the direction of maximum stress. The single grain having a different orientation gives evidence of having undergone greater stresses than other biotite grains. It shows strain shadows which mark two sets of intersecting fractures. Each set cuts the long axis of the grain at an angle close to 50 deg., so that the 100-deg. obtuse angle lying between them is bisected by the long axis of the mineral grain. This grain appears to have failed as a result of stress applied along the long axis,

Fig. 7—A large grain of albite. Crossed nicols. Mag. 20.

This grain has been broken and displaced by movement along a fracture in the specimen. Fragments of the grain have been dragged along the fracture.

A large grain of albite in one speci-

men is not only fractured, but is badly mashed. The major fractures are in two sets and are marked by streaks of crushed material rather than by clean breaks. Many of the larger fragments lying between these streaks are in the form of strain-rhombs, which indicate the direction of the rotational stress that fractured the specimen.

In the specimen shown in Fig. 6 the direction of movement is shown by the elongation and inclination of the major axis of lenticular patches of quartz and feldspar. Areas of these minerals have been squeezed, drawn out, and tilted by the movement of the rock mass as a whole, and indicate the relative movement of the various parts. In the illustration the upper part of the field has moved to the right relative to the lower part.

Fig. 9—Looking eastward along the fault

Trace of the fault plane extends from the lower right corner upward to the upper left part of the picture. The rock wall above this line is the fault surface on the hanging wall. It dips away from the observer and to the right at an angle of 63 deg. The hanging wall relative to the footwall has moved upward and to the rear of the view.

In some specimens, larger grains or crystals have undergone drag effects. One instance of this is shown in the specimen illustrated in Fig. 7, where a large grain of albite lies along a fracture. The grain was crushed and fragments were dragged along the plane of movement. The drag, of course, indicates the direction of movement.

Other effects of the rotational stresses acting along the faults which help to determine the relative movement are strain-rhombs in the rock mass, if larger single grains are not present; and overturned folds in streaks or bands following earlier fracture or shear planes.

The specimens already described checked with the field observations on the direction of movement and relative movement of adjacent blocks of the fault under consideration. The lower part of Fig. 8 is a three-dimensional drawing of the plan in the upper part of the figure and is an attempt to show the relations of part of the No. 2 level and two faults which intersect it as illustrated.

The tabular orebody strikes east-northeast roughly parallel to the general trend of the drift. It dips southeastward at an angle of 55 deg. and plunges northeastward at an angle of about 18 deg. On the three-dimensional block the orebody would strike east-northeast across the top and have an eastward apparent dip on the front or south face.

The specimens discussed were taken from the three positions marked "X" along the more southerly of the two faults—that is, the one dipping 63 deg. southward. The thin-sections were cut, so as to be perpendicular to the fault plane and roughly parallel to the striæ on the fault plane which are inclined to the dip, indicating strike-slip as well as dip-slip movement. (See Fig. 9.) The evidence of the thin-sections as well as of the fault surface indicates a thrust fault with the south block riding upward and eastward relative to the north footwall block.

To determine the direction of movement along a fault from grooving, striations, and other evidence is usually a simple matter, but to ascertain whether the hanging wall moved to the right or left or up or down is not always so simple. Oriented thin-sections of specimens advantageously chosen should be of considerable help in solving the latter problem.

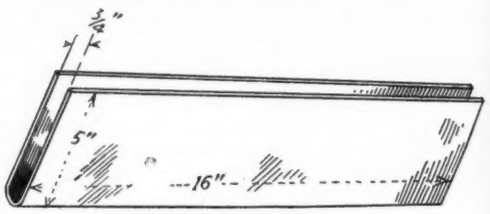
I wish to express my appreciation to Mr. R. E. Crockett, superintendent, Alan Wood Mining Company, for allowing access to the mine and to certain maps from which the plan shown in Fig. 8 was taken; also to Mr. John Holmes, mining engineer, for assistance in underground investigations.

Design of Automatic Sample Cutters

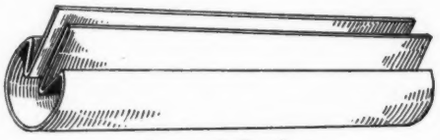
TWO automatic samplers of the triangular box type, operated by water, were installed in a small pilot mill, constructed on a gold property for the purpose of determining the advisability of erecting a larger mill, writes S. R. Moore, Marysville, Mont. The ore was a pure quartz carrying a small amount of sulphide. About 90 per cent of the valuable mineral content was in gold and 10 per cent in silver. About 60 per cent of the gold was free and fine. Flotation was employed, the ratio of concentration being 100 to 1.

The samplers, one for the heads and one for the tailing, each cut the stream about six inches below the discharge from a launder, the positions of the samplers being identical with respect to the two launders. Both samplers were set to operate every 2½ min., and the only difference noticeable at the two points was that pulp density of the tailing was much lower than that of the heads. The cutters, made of galvanized iron and similar to those used elsewhere, were identical, their dimensions being 16 in. long, 5 in. deep, and ¾ in. wide. To determine whether splashing prevailed was difficult, but undoubtedly it occurred at the tailing sampler.

Hand sampling was employed to check the automatic sampling. The automatic tailing samples were 50 to 300 per cent higher than the hand samples. In two instances the tailing samples were higher than the head samples. Calculations indicated clearly that the error was in the automatic tailing sample, and a careful check showed that the samples were not being accidentally salted. The two head samples invariably checked closely.



Incorrect Design



Correct Design

After two weeks' trial on two shifts daily, the design of the sample cutters was changed, as indicated in the accompanying sketch. Thereafter, close checks were obtained. Just why a heavy concentration of the valuable mineral constituents of the ore should occur in the tailing and not in the heads is not readily understood. The thinner tailing pulp may account for it.

Hoist Design and Construction

PART II

Lucien Eaton

FIVE qualities of wire rope are in commercial use—Swedish iron, cast steel, extra-strong cast steel, plow steel, and improved plow steel, the tensile strength and elastic limit of the wire increasing in the order named. Four constructions are standard: 6x7, 6x19, 6x37, and 8x19. The usual construction for ordinary hoisting conditions is 6x19; that is, six strands of nineteen wires, each wrapped about a hemp core. The rope may be regular lay, in which the strands are twisted to the right and the wires to the left, or Lang lay, in which both strands and wires are twisted to the right. In the 6x7, 6x19, and 8x19 constructions the wires are normally all of one size; but in the 7x37 construction and in the Seale patent constructions, often used in Lang-lay ropes, are three different sizes of wires. Large hoisting ropes, 2 in. or more in diameter, are often of 6x37 construction, to avoid the use of too large wires. Lang-lay ropes are preferred where wear is mostly frictional, as in inclined shafts; but, because of their tendency to untwist, they should not be used where the end is often disconnected. In recent years ropes having pre-formed wires and strands have been constructed, and these have no tendency to untwist, whether regular or Lang lay. Lang-lay ropes of 6x9 construction are especially good for use in incline shafts, on account of the large wires used in their construction. Some ropes are too stiff to be suited to the usual drums and sheaves; others are flexible enough for all ordinary requirements.

On account of their greater strength and durability, plow-steel and improved plow-steel ropes are gaining in favor rapidly; and, in spite of their higher cost, are more economical in service than the softer grades. On account of their higher elastic limit they can be used under more severe bending conditions than cast-steel or iron ropes.

In determining the size and quality of rope to be used, two methods are in general use. If the diameters of the drums and sheaves are generously large, weight alone may be taken into consideration. A factor of safety of 5 is usual for skips, but for cages or skips that handle men it should be 7. In calculating the gross load, the weight of the full length of suspended rope is added to the weight of the loaded skip or cage, and the stress due to acceleration is added to the sum. If, however, the sizes of the drums and sheaves are near the allowable limit, bending stresses should be taken into account. When these are added to the gross load, a lower factor of safety is used. The stress due to bending is calculated by the formula:

$$K = 733{,}000\frac{D^2}{S + D},$$

where S is the diameter of the sheave or drum in inches, D is the diameter of the rope in inches, and K is the total, not unit, bending stress. When bending stress is added to gross load, a factor of safety of 3 corresponds to the factor of 5 for gross load alone, and is found by experience to be sufficiently large. That size and quality of rope should be used the breaking strength of which, as given in the manufacturer's table, is next above the amount obtained by multiplying the sum of the gross load and bending stress by the factor of safety. The breaking strengths of ropes given in the catalogs may safely be used, and are more reliable than calculations by formula. The strength of a wire rope increases during the first three or four months of use, and then decreases as the outer wires become worn. No definite rule has been formulated and generally adopted for the limit of allowable wear, before a rope is discarded, but the following rule is sometimes used: "A rope should be discarded when the outer wires are worn more than halfway through, or when more than 10 per cent of the wires in one full lay are broken, or when the diameter or circumference has decreased more than 10 per cent." Wires worn thin indicate abrasion. Wires broken off squarely indicate drums or sheaves of too small diameter. Corroded wires indicate lack of lubrication or acidity in the mine water, or both.

Care and inspection of hoisting ropes are important. Each rope should be inspected casually every day and carefully once a week by a competent man, and a detailed record should be kept. Several feet of rope at the end nearest the skip should be cut off periodically, at intervals of three to six months, and the rope should be turned end for end, when it has been in use two-thirds of its expected life. A form of record recommended by the American Society of Mechanical Engineers is shown herewith. To this record should be added the dates when the rope was put on, turned end for end, and taken off, and the dates when pieces were cut off, and the amount so cut off. The name of the mine and the particular shaft, skip, and cage should also be noted. The hoisting speed should be taken into consideration when making comparisons.

Special greases or compounds have been prepared for the protection of wire ropes from acid waters and for reducing friction. They penetrate to the core only when hot, and are difficult to apply. No lubricant is effective in very cold weather, but except at extremely low temperatures a light oil gives good general results. It is easily applied by allowing it to drip through a petcock from a bucket hung over the head sheave. Waste oil is ordinarily used for this purpose.

The best attachment for the end of the rope is a tapered socket made of wrought iron. The wires at the end of the rope are separated, cleaned first with gasoline and then with dilute muriatic acid, and are inserted in the socket, which is then filled with pure melted zinc. Such a socket develops the full strength of the rope. A thimble spliced in the end of the rope is sometimes used, and its efficiency varies from 75 to 90 per cent. Probably the most common attachment is a thimble, around which the rope is fastened with wire-rope clips. This form of attachment, if properly made, has about the same efficiency as the splice. Improperly made clip attachments have efficiencies as low as 60 per cent. Not fewer than four clips should be used, one lay apart, and the U-bolt should always be placed on the dead end of the rope. The clips should be tightened after one hour's run and at regular inspection intervals afterward.

The tonnage that may be hoisted per

Rope-Service Data Sheet

Description and Service:
Rope No. ..... Manufacturer .....
Size ..... Material ..... Construction .....
Rated Strength (Breaking Stress): Rope ..... Wire .....
Type of Application .....
Period of Service ..... Loading .....
Mileage (Total During Service) ..... Height of Rise .....
Number of Bends ..... Number of Reverse Bends ..... Sheave Diameter .....
Acceleration and Deceleration .....
General Remarks (Regularity of lay; corrosion conditions; lubrication, type, and condition of sheave grooves by gaging; shackling.) .....
.....

This is one of a series of articles on mine equipment.

shift can be calculated from an analysis of the hoisting cycle. In practice, the actual time during which hoisting is carried on in an 8-hour shift is from 7 to 7½ hours, and 7 hours should be taken as the basis of calculation, and 20 per cent deducted from the theoretical capacity as an allowance for incidental delays. With well-designed skip-loading equipment the loading time should not exceed 5 seconds, and it will be safe to use 10 seconds. Some time also must be allowed for dumping the skip, and 10 seconds is a safe allowance. The time required for acceleration and deceleration depends on the weight of the equipment and the maximum hoisting speed. Rates of acceleration vary from 1.5 ft. to 13 ft. and normally are from 3 to 5 ft., all per second per second. If acceleration and deceleration take place at a uniform rate, the average hoisting speed during these periods will be half the maximum speed, and the distance traveled by the skip can be calculated accordingly. The length of the acceleration period is usually 10 to 15 seconds, and at moderate speeds the rate of deceleration is approximately the same. At high speeds, however, deceleration is often at a slightly lower rate. If the size of the skips, the rope speed, and the rate of acceleration or period of acceleration are known, the hoisting cycle and the capacity per shift are easily calculated. If the actual capacity is much less than the theoretical, unusual delays can usually be found to account for the discrepancy. By using the same sort of calculations, the speed and equipment necessary to obtain a given output can be determined by trial and error.

As an example let us assume two 5-ton skips in balance, a rope speed of 1,500 ft. per minute, acceleration and deceleration periods of 10 seconds, and a hoisting distance of 1,200 ft., including the depth of the loading point below the bottom level, and the height of the dumping point above the collar of the shaft. The figure used would be approximately that for a level 1,000 ft. below the collar. To reach a speed of 1,500 ft. per minute, or 25 ft. per second in 10 seconds, requires an acceleration of 2.5 ft. per second per second, which is conservative. The average hoisting speed during the acceleration and deceleration periods is 750 ft. per minute, and the distance traveled in each period is 125 ft. This leaves 950 ft. for the skip to travel at full speed, and this will take 38 seconds. The hoisting cycle will therefore be as follows: Loading, 10 seconds; acceleration, 125 ft., 10 seconds full speed, 950 ft., 38 seconds; deceleration, 125 ft., 10 seconds; dump, 10 seconds; total 78 seconds.

One hour is 3,600 seconds, and the number of cycles per hour will therefore be 46, and the output per hour at 5 tons per trip will be 230 tons. Deducting 20 per cent for incidental delays, gives a capacity of 184 tons per hour. Seven hours of hoisting will give 1,288 tons as the average capacity per shift, including waste and ore. In planning equipment for the disposal of product, however, one must remember that, at certain periods, production will be at the rate of 230 tons per hour.

The calculations for determining the horsepower of the hoist motor are elaborate, and include an analysis of the power required during each of the periods of the cycle and a graph or hoist-duty cycle diagram. Motor rating is based on the load carried for a given time with a given rise in temperature. On a basis of one hour and 40 deg. C. temperature rise, a motor rated at 1,000 hp. should carry that load continuously for one hour without a rise of more than 40 deg. C. in the temperature of its windings. Motor temperature varies approximately as the square of the current and directly as the time. To determine the motor capacity, the "heating load horsepowers" for the three periods of the cycle are averaged. The average horsepower for each period is then squared and multiplied by the time of its duration, and the sum for the three periods is then divided by the total time. The square root of the quotient gives the horsepower required. These calculations are given in considerable detail in the "Goodman Mining Handbook" (fifth edition, 1927) pp. 150-164.

The manufacturers usually make these calculations, but under stress of competition they are likely to be too optimistic in regard to motor capacity, and a simple method of checking their results is desirable. A simple empirical formula, for which I am indebted to Mr. O. D. McClure, chief mechanical engineer of Cleveland-Cliffs Iron, at Ishpeming, Mich., is based on experience, and gives excellent results for hoists up to 1,000 hp. By this formula the alternating-current motor size is double the horsepower required when the skips and ropes are in exact balance and are traveling at full speed at the middle of the cycle. In the example quoted the calculation would be as follows:

$$\text{Horsepower} = \frac{\text{Wt. of ore in lb. x speed in ft. per min. x 2}}{33{,}000} = \frac{10{,}000 \times 1{,}500 \times 2}{33{,}000} = 909.$$

On account of its better cooling qualities, a direct-current motor need be only 0.7 as large as an alternating-current motor, or in this case 636 hp. Abnormally high rates of acceleration or great depths of shaft would have to be given special consideration when this formula is used.

The hoisting cycle and horsepower requirements for cages are similarly calculated, but the hoisting speed and rate of acceleration are not usually as high when men are carried, and plenty of time must be allowed for getting the men on and off the cage. One second per man should be ample for loading and unloading the cage, and 5 to 10 seconds must be allowed for closing the door and repeating the signals. Cars should be taken on or off the cage in 15 to 20 seconds. In calculating the horsepower of the motor the largest weight that will have to be taken into the mine, such as a locomotive, must be used, and sufficient torque must be provided to move it.

The number of men that can be carried on a cage depends on the floor area and the type of inclosure. Fully inclosed cages permit greater crowding, and one man for every 1½ sq.ft. of floor area is permissible, but if the cage is not fully inclosed, 1¾ or even 2 sq.ft. of area are required. Weight per man varies considerably in different localities. In Rhodesia, 125 lb. for native "boys" is sufficient, and in the Lake Superior country 180 lb. is not too large. An average of 170 lb. may safely be adopted.

Composition of Copper Matte

C. K. Avetisian

COMPLETE ANALYSES of compounds formed by copper matte and slag, to determine the iron, sulphur, and oxygen, is an important aid in obtaining correct furnace control in copper smelting. This question, although studied by many investigators, is far from exhausted, and many controversies and conflicting data remain to be settled. Comparatively few have employed chemical methods to determine the principal compounds of the matte; most work in studying matte structure has been by metallographical methods, and a number of contradictory equilibrium diagrams, based on the binary system $Cu_2S.FeS$, have been plotted. The diagrams by Röntgen[1], Hofman[2], Baïkov and Trutnev[3], Bornemann and Schreyer[4], Carpenter and Hayward[5], are all well known; the diagram of the ternary system CuFeS, studied by Reuleaux[6] in the area of the quadrangle Cu_2S-FeS-Fe-Cu, should also be mentioned, as well as the microscopic examinations by Fulton and Goodner[7].

Copper matte has long been considered a mixture of the simplest sulphides Cu_2S and FeS, the only ones stable at high temperatures. In the course of time, however, a deviation from this composition was noticed, the sulphur content being insufficient to form Cu_2S and FeS. Investigation of magnetic properties led to the conclusion that a substantial amount of metallic iron was present. Microscopic examinations, however, did not confirm this assumption, and the suggestion was made by Peters, Richards, and Hofman, that iron oxide was contained in the matte. Professor Mostovich[8] proved that oxygen is always present in matte, as magnetite or ferrite. Prof. Baïkov[9] states that oxygen may be present only as ferrous oxide, as FeS and SO_2 reduce Fe_2O_3 and Fe_3O_4.

Depending upon the rate of cooling, matte may undergo considerable change, for instance in specific gravity; sometimes metallic "moss" copper is formed. Some investigators assume cooling to be accompanied by liberation of metallic copper from sulphur compounds and by formation of higher sulphur compounds of copper and iron. Röntgen claimed the formation of an iron sulphide with higher sulphur content than FeS; Baïkov, by metallographic investigation, pointed to formation of $CuFeS_2$. In their classical work on the composition of matte, published in 1906, Allan Gibb and P. C. Philp[10] dissolved all copper compounds in a neutral silver nitrate solution, in which reagent FeS is insoluble and metallic iron only slightly soluble. Examination has shown metallic iron to be absent in commercial matte; metallic copper was detected, however, soluble in an excess of FeS, as against $5Cu_2S.FeS$, the only compound determined by Gibb and Philp and designated "the white metal." With few exceptions silver nitrate solution recovered all of the copper. Later on, however, investigations by Mostovich, Wartman, and Bayer[11] demonstrated that silver nitrate is ineffective in dissolving copper sulphide. Thus, doubt is cast on the value of the work of Gibb and Philp, who deny the presence of magnetite in the matte because treatment with silver nitrate did not leave a residue of magnetite. With few exceptions, all mattes treated by them were high in copper, from 49.7 to 80 per cent, a condition which might have had great influence upon their conclusions. Howley, however, in his subsequent work[12] demonstrated that the amount of magnetite is proportional to FeS or inversely proportional to the copper content of the matte.

Gibb and Philp, melting copper with iron sulphide, obtained a lower layer of metallic copper and an upper layer of matte, analyzing 60.6 per cent copper, 17.8 iron, and 21.6 per cent sulphur. Baïkov and Trutnev, in 1908, melting iron with cuprous sulphide, obtained a metallic layer and a matte with 58.87 per cent copper, 17.07 iron, and 20.99 per cent sulphur. Comparing their results with those of Gibb and Philp, they came to the conclusion that the reaction $Cu_2S + Fe = FeS + 2Cu$ is reversible, the equilibrium concentration at melting temperature corresponding to a matte of an almost similar composition, FeS and Cu_2S.

In 1911, Professor Yushkevich[13] studied powdered mixtures of Cu_2S and FeS; Cu_2S and Fe; FeS and Cu; and FeS and Cu_2O, to establish the reactions between sulphide and oxide compounds of iron and copper. The mixtures were heated below the melting temperature, in an atmosphere of nitrogen, and a complete analysis was made of the resulting products. Dissolving the products in 7 per cent potassium cyanide in a hydrogen atmosphere, he came to the conclusion that a single compound, $(Cu_2S)_3FeS$, was formed by heating above 200 deg. C., in amounts increasing with the temperature. The same compound was detected by Borneman and Schreyer by microscopic examination, and was considered the most stable of all compounds in matte.

According to Yushkevich, potassium cyanide dissolves copper, cuprous sulphide, and cuprous oxide; iron and ferrous oxide are dissolved slowly, and ferric sulphide is insoluble. In spite of this, a considerable amount of FeS was found to dissolve from mixtures of FeS and Cu_2S after heating from 200 to 800 deg. C., the amount dissolved increasing with the temperature. Upon analysis, Yushkevich found twice as much sulphur combined with copper as with iron, pointing to the compound $(Cu_2S)_3FeS$. Instead of the reversible reactions between copper, iron, and their sulphides mentioned by Trutnev and Baïkov, Yushkevich suggested the following reactions: $3Cu_2S + Fe = (Cu_2S)_3FeS + 2Cu$ and $4Cu + 3FeS = (Cu_2S)_3FeS + 2\,Fe$.

In 1925, Mostovich[8] investigated the method of dissolving artificial and commercial matte in silver nitrate. At the same time he determined oxygen in the samples by reduction with carbon monoxide and hydrogen at 820-830 deg. C. At higher temperature the matte would sinter and incomplete reduction result. As he found the reduction with carbon monoxide incomplete at this temperature, he adopted the reduction with hydrogen, according to Hampe. Neutral solution with 32 grams silver nitrate per liter was stirred mechanically, at room temperature, in a sealed dark flask, with the following finely ground materials: copper; iron sulphide, obtained by calcining pure pyrite in nitrogen at 1,000 deg. C.; cuprous sulphide made by fusing copper and sulphur in graphite crucibles; natural and fused chalcopyrite; and several laboratory and commercial mattes.

The experiments showed complete solution of copper in 4 hours. FeS did not dissolve at all, and Cu_2S slowly, 86.6 per cent dissolving in 4 hours and 98.9 per cent in 24 hours. Only 9.5 per cent copper dissolved from the natural chalcopyrite, 40.85 per cent from the fused material, with agitation of 28 hours in both cases. Neither commercial matte nor laboratory matte showed complete solubility of copper, 80 to 95 per cent being dissolved, the amount of insoluble copper increasing with the iron content; this points to stable insoluble compounds of FeS and Cu_2S. Not obtaining satisfactory results with a 7 per cent potassium cyanide solution, Mustovich doubted the Yushkevich theory about the formation of $(Cu_2S)_3FeS$, and he rejected this solvent for determining iron and copper compounds in matte. By determining oxygen and reverting to the simplest copper and iron compounds, stable at high temperature, he found that in some samples oxygen was combined with iron in greater amounts than re-

quired by the oxide Fe_2O_3, which indicated that sulphides higher in sulphur than Cu_2S and FeS were present. Studies of magnetic permeability and comparison with oxygen determinations led Mostovich to the conclusion that oxygen, present as ferrite, $Fe_2O_3.FeO$, was an integral component in copper mattes, blast-furnace mattes as well as reverberatory mattes, and on this basis he presented the "ferrite" theory of pyritic smelting.

The amount of magnetite in commercial matte has recently been studied by Hawley[12]. He dissolved matte in a mixture of sodium chlorate, nitric acid, and sulphuric acid, in which magnetite does not dissolve as readily as the insoluble impurities, and assumed the difference in weight of insoluble impurities and insoluble residue to be magnetite, Fe_3O_4. On this assumption he based his figures for the oxygen content of the matte, figures which are considerably lower than those of Mostovich. Hawley also concluded that a constant ratio, ranging from 0.12 to 0.55, existed between magnetite and FeS, more magnetite being found in matte of low copper content. The same conclusion was drawn from Mostovich's work on oxygen content in matte.

A brief review of the literature shows that no method has yet been found which gives a rational, complete analysis of mattes. In studying the solvents used to separate the various compounds, I selected 7 per cent potassium cyanide solution, the concentration used by Yushkevich and Mostovich. Tests with concentrated solution were discontinued on account of the difficulties is analyzing the leach solution. The solubility of the following compounds was determined at room temperature: Copper, cupric sulphide, cuprous and cupric oxide, copper ferrite, chalcopyrite, ferric sulphide, and ferric oxide. Tests were made on matte of varying copper content.

Electrolytic copper, 0.5 gram sample, in the form of fine wire dissolved very slowly; fine shavings, made with a steel file, with all iron subsequently removed with a magnet, dissolved completely after five hours' agitation. A one-gram sample of Kahlbaum's Cu_2O dissolved completely in one hour, and one gram of Merck's pulverized Cu_2S in two hours as well as precipitated CuS. A 0.1539 gram sample of CuO made by calcining Merck's Cu_2O was treated for two days, 99.84 per cent of the copper dissolving.

FeS was obtained by fusing a pyrite containing 45.6 per cent iron, 52.3 sulphur, 2.5 silica, and 0.12 per cent copper. A half a gram sample was treated for two days, with only a trace of iron going into solution. No iron dissolved from a sample of Kahlbaum FeO treated for five days. Pure grains of chalcopyrite, $CuFeS_2$, selected from Zangezur ore, when ground to minus 325 mesh, showed some soluble iron, copper, and sulphur from oxidized sulphates. A coarser sample, with grains ranging from 0.5 to 3 mm., gave only traces of these elements in solution after six hours' agitation.

Copper ferrite, $CuO.Fe_2O_3$, was prepared by fusing stoichiometric amounts of finely ground Merck's CuO and Fe_2O_3 in a porcelain dish in a Tamman electric furnace. The fusion thus obtained was magnetic and of uniform appearance. A one-gram sample, ground to below 325 mesh, was agitated for twelve hours, after which time only 1.29 per cent of the sample had dissolved. The dissolved copper and iron may have occurred in the free or reduced state, instead of as combined ferrite.

The results of these experiments, and the results obtained by previous investigators, are compiled in Table I. They show that a 7 per cent potassium cyanide solution at room temperature has a definite effect on several compounds of iron and copper found in metallurgical products. The results are in accord with previous investigations, and invite no such controversies as do those obtained with silver nitrate. Copper and all its compounds, with the exception of $CuFeS_2$ and $CuO.Fe_2O_3$, are completely soluble in cyanide, whereas iron and its compounds are insoluble. As observed by Yushkevich and Mostovich, ferrous sulphide heated with cuprous sulphide is partly dissolved in cyanide, with the fixed cuprous sulphide. The composition of the copper-iron compound dissolved may therefore be ascertained by determining the amount of sulphur dissolved in addition to the copper and iron in solution. Assuming sulphur to be present as K_2S, the two investigators mentioned precipitated it with mercuric cyanide, $Hg(CN)_2$. Special care must be exercised in this work to avoid loss of sulphur. The following equations are involved:

(1) $2Cu_2S + 4\,KCN + 2H_2O + O_2 = Cu_2(CNS)_2 + Cu_2(CN_2) + 4KOH$

(2) $Cu_2(CN)_2 + 6KCN = Cu_2(CN)_2 \cdot 6KCN$

(3) $Cu_2(CNS)_2 + 8KCN = 2KCNS + Cu_2(CN)_2 \cdot 6KCN$

(4) $Cu_2S + 8KCN = Cu_2(CN)_2 \cdot 6KCN + K_2S$

Yushkevich and Mostovich both admitted that dissolution of Cu_2S follows only Equation 4; dissolution of sulphides usually involves Equations 1, 3, and 4. Oxygen, which is required in reactions 1 to 3, is generally dissolved in water and in the cyanide solution. Yushkevich has effected dissolution and filtration in an atmosphere of hydrogen, to prevent oxidation of the K_2S formed. If air be admitted, K_2S is decomposed, yielding elementary sulphur which passes into the residue: $K_2S + H_2O + \frac{1}{2}\,O_2 = 2KOH + S$. With carbon dioxide in the air sulphur may form hydrogen sulphide: $K_2S + H_2O + CO_2 = H_2S + K_2CO_3$. Both reactions were observed in dissolving Cu_2S in cyanide. A yellow slime left in this process, when oxidized with chlorate and nitric acid, yielded a white precipitate with barium chloride. When the dissolution was conducted in a sealed vessel through which a stream of air was passed, the air leaving the vessel did not blacken Dreksel absorbers filled with mercuric cyanide and lead nitrate during 24 hours' dissolution. Upon addition of carbon dioxide, however, the reagents were rapidly turned to a black color by the hydrogen sulphide evolved. If oxygen is present some Cu_2S dissolves according to Equation 1. Sulphur dissolved may dissolve as KCNS and as K_2S. The former does not react with mercuric cyanide, the reagent used by Yushkevich and Mostovich for determination of dissolved sulphur. To determine the distribution of sulphur in these two compounds two one-gram samples of white matte from the Karabash and Kalata smelters were leached, containing the following percentages, respectively: Cu, 80.06; Fe, 0.36, S, 19.62; and Cu, 71.89; Fe, 4.78; S, 22.36. Sulphur, combined as K_2S, was first precipitated with mercuric cyanide and the precipitated HgS filtered off; the sulphur in the filtrate, combined as KCNS, was then oxidized with nitric acid and chlorate and precipitated as barium sulphate.

Results:

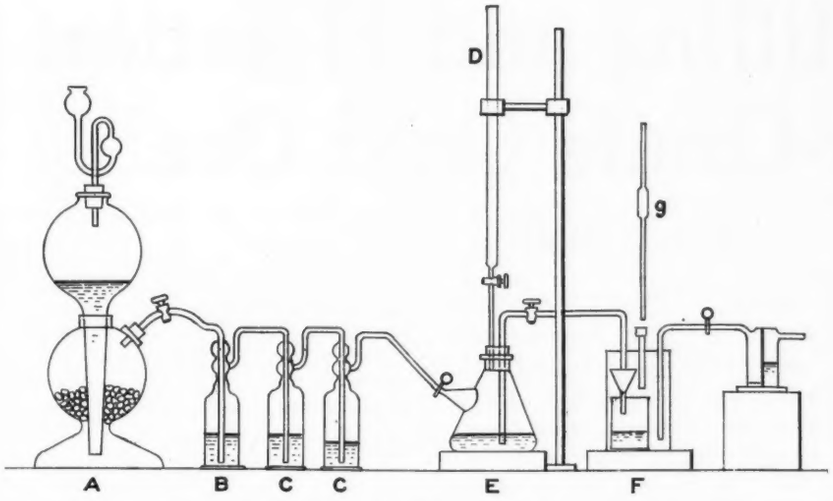
	Karabash Matte	Kalata Matte
Ppt. as HgS, grams S	0.1711	0.1910
Ppt. as $BaSO_4$, grams S	0.0258	0.0311
In residue, grams S	0.0007	0.0027
Total, grams S	0.1976	0.2248

Results obtained from the experiment showed that no loss of sulphur occurred when the sulphides were dis-

Table I—Solubility of Copper and Iron Compounds in 7 Per Cent KCN Solution in an Atmosphere of Hydrogen

Compound	Observation by Yushkevich	Mostovich	Wanjukoff (14)	Author
Copper	Complete	Complete		Complete
Cu_2O	Complete		Soluble	Complete
CuO			Soluble	Complete
$CuO.Fe_2O_3$		Insoluble		Insoluble
Cu_2S	Complete	Complete	Complete	Complete
CuS				Complete
$CuFeS_2$		Insoluble		Insoluble
Iron	Very slowly			
FeO	Very slowly	Insoluble		Insoluble
Fe_2O_3		Insoluble		
Fe_3O_4		Insoluble		
FeS	Insoluble	Insoluble	Insoluble	Insoluble
FeS_2		Insoluble		
Fe_nS_{n+1}		Insoluble		

Tests by Wanjukoff: saturated solution.



Apparatus used in dissolving matte in potassium cyanide solution, in hydrogen atmosphere. A, Kipp apparatus for production of hydrogen. B, Absorption flask with potassium hydroxite. C, Absorption flask with mercuric cyanide. D, Burette with 7 per cent potassium cyanide. E, Absorbtion flask. F, Filter. G, Pipette for sampling filtrate.

solved in a hydrogen atmosphere, and that the ratio of sulphur in K_2S and KCNS was 87:13 for Karabash matte and 86.4:13.6 for Kalata matte. Sulphur remaining in the residue is not of secondary origin, precipitated by oxidation of K_2S. In both analyses it contained a corresponding amount of iron. A partial conversion of sulphur into sulphocyanide compounds during solution in an atmosphere of hydrogen was thus proved. In varying ratios it occurred in all experiments. A satisfactory sulphur balance was always obtained. How can the absence of sulphocyanides be explained in the experiments by Yushkevich and Mostovich, who do not mention this subject? Working with pure Cu_2S, Yushkevich precipitated sulphur as HgS, never noticing the lack of sulphur. Of a total of 101.5 mg. of sulphur in his sample of Cu_2S, Mostovich precipitated 99.25 mg. as HgS. Only 2.2 per cent was unaccounted for, a figure close to the limit of analytical error. The only difference in my experiments was the use of mechanical agitation in an atmophere of hydrogen, which could not have vitiated the results.

To throw light upon this subject two 0.5 gram samples of Merck's Cu_2S were dissolved in an atmosphere of hydrogen in 100 c.c. cyanide solution which had been boiled for 20 minutes, then cooled and poured into a burette. Precipitation of sulphur as HgS gave 18.64 and 19.34 per cent dissolved sulphur in both tests, whereas the original sample contained 79.86 per cent copper and 20.14 per cent sulphur, a loss of 1.5 and 0.8 per cent sulphur in the two experiments—sulphur which had passed into sulphocyanide.

Formation of KCNS takes place by action of polysulphides on potassium cyanide, or by boiling this reagent with sulphur. Pyrite and markasite may also form KCNS. The presence of dissolved oxygen is not required in any of these instances. As KCN is a reducing agent, CuO and CuS dissolving in KCN must form KCNO and KCNS, respectively. In the presence of K_2S, KCNO will be converted into KCNS; dissolution of CuS in KCN results in the formation of KCNS, thus: $2CuS + 9KCN = Cu_2(CN)_2{\cdot}6KCN + K_2S +$ KCNS. To demonstrate this, a sample of precipitated copper sulphide was dissolved rapidly in potassium cyanide. A fine yellow slime appeared, which dissolved in 2 to 3 minutes, showing that the reaction passes through the following stages: (1) $2\ CuS + 8\ KCN = Cu_2(CN)_2{\cdot}6KCN + K_2S + S$; (2) $KCN + S = KCNS$. Formation of KCNS is assured by the presence of CuO or CuS. The experiments were made with samples of Kalata blast-furnace matte, with 23.27, 23.85, and 24.65 per cent copper, and Korsak-Pai converter matte, with 70.17 per cent copper, and reverberatory-furnace mattes from Karsan-Pai with 50.2 and 62.84 per cent copper. The mattes were ground in agate mortar to minus 325 mesh, and two samples were taken from each lot. One of these was leached with KCN; the other was used for the oxygen determination, which was made by the following method:

Hydrogen, produced in a Kipp apparatus, was passed through mercuric cyanide to absorb hydrogen sulphide and through sulphuric acid to remove moisture. After passing through a tube with calcium chloride it was led through a heated glass tube, filled with copper shavings, where oxygen was removed. Finally it passed through absorption tubes filled with calcium chloride and phosphor pentoxide, and a glass-wool filter, whereupon it entered the porcelain tube of a Heraeus combustion furnace, in which was placed a porcelain boat with the matte sample. Leaving the furnace, the hydrogen passed through a Lunge absorber filled with a mixture of bromine and hydrochloric acid, then through a Drexel absorber, containing the same mixture, and finally through an absorber filled with potassium hydroxide, where the bromine hydrochloric acid fumes were absorbed. The temperature of the furnace was measured with a platinum-rhodium thermocouple. In making the determination the boat was filled with matte, hydrogen was passed through for about 1 to 1½ hour, power was turned on, and the furnace slowly brought up to 820-830 deg. C., as recommended by Mostovich. This temperature was maintained for about two hours, and the furnace was then slowly cooled to room temperature, maintaining the flow of hydrogen. Mostovich figured the oxygen content by subtracting the sulphur absorbed in the mixture of bromine and hydrochloric acid from the loss in weight of the boat, a method formerly also used by me. No sintering was observed at the temperature used. Leaching was done with matte ground to minus 325 mesh, in an atmosphere of hydrogen, with violent agitation for at least six hours, followed by filtration in an atmosphere of hydrogen. The apparatus used is shown in the accompanying sketch. Dissolved sulphur was determined both as sulphide and as sulphocyanide. Oxygen was also determined in the residue except for mattes containing 55.2 per cent copper or more, which gave only a negligible amount of residue. Finally the reduced residue was analyzed for copper, sulphur, and iron. Thus the distribution of the constituents of the matte could be definitely established.

Bibliography

1. *Metallurgie,* Vol. 3, p. 479 (1906).
2. Transactions A.I.M.E. Vol. 38, p. 424 (1908).
3. Transactions Petersburg Polytechnic Institute, Vol. 10, I, p. 309 (1908).
4. *Metallurgie,* Vol. 6, p. 619 (1909).
5. *Engineering and Mining Journal-Press,* Vol. 115, p. 1055 (1923).
6. *Metall und Erz,* Vol. 5, p. 99 (1927).
7. Transactions A.I.M.E., Vol. 39, p. 584 (1908).
8. Transactions Siberian Institute of Technology, Vol. 1 (1925).
9. Journal of Russian Metallurgical Society, Vol. 1, 1925.
10. Transactions A.I.M.E., Vol. 36, p. 665 (1906).
11. *Metal Industry* (London) Vol. 36, No. 19, p. 504 and No. 21, p. 553 (1930).
12. *Engineering and Mining World,* Vol. 2, No. 5, p. 270 (1931).
13. Journal of Russian Metallurgical Society, No. 5, (1911).
14. W. A. Wanjkoff: To the question of the affinity of elements at high temperature (Russia), p. 87, (1916).

(To Be Continued)

Gravity Milling and Flotation of Low-Grade Gold Ore

John B. Huttl

An example of the successful combination—of a new crushing principle, the Hadsel; with modern flotation practice, reagents, and Kraut cells; and leaching with cyanide solution—in the economical treatment of a comparatively small daily tonnage of a low-grade gold ore.

ONE of the most interesting mine surface plants to be found in the West is that of the Beebe Gold Mining Company, in Georgetown, Eldorado County, Calif., about 22 miles east of Auburn. Milling practice involves features that are new in the United States and not common elsewhere. The results achieved in the mining of a low-grade orebody, and a remarkably simple metallurgical flowsheet, afford an object lesson that will be of considerable value in the development of gold mines. Despite the attention the property has received locally, little regarding it has appeared in print, so the following description may find readers to welcome it. A short sketch of mining operations is added. The data presented were obtained on a recent visit to the property, made possible through the courtesy of Alex Wise, president, and F. A. Wise, general superintendent, of the company.

The Beebe ore deposit is a vein of silicified schist contacting a slate belt on the hanging wall and a diorite dike from 1 to 2 ft. thick on the footwall. Thickness of the vein varies up to 50 ft. The strike is about N. 30 deg. E., and the dip 75 to 80 deg. E. Two kinds of ore are being mined from the deposit: a highly silicified schist which is very hard and tough, and a softer grade composed of more schist than quartz. About one-third of the gold in the ore is finely grained free gold; the remaining two-thirds is associated with pyrite. Average gold content is about $3 a ton.

Early mining was done by glory-hole methods; shrinkage stoping is used by the present management. The mine is served by a vertical shaft 250 ft. deep, containing a hoisting compartment and a manway measuring 4½x4 ft. and 4½x2 ft., respectively, inside the timber. Equipment includes a steel headframe and a single-drum hoist fitted with a G. E. Thrustor automatic brake. Hoisting speed is 400 ft. a minute. All underground work is confined to the 130-ft. level, north of the shaft. Exploration drifts measure 5x7 ft. in cross-section. No timber is used, as the ground requires no support. Drilling is done with IR-L 74 drifters, using 1-in. hexagonal, hollow steel. The standard round consists of sixteen holes, drilled 5 ft. deep. Hercules 40 per cent dynamite is used for blasting.

The raises, measuring 6x10 ft. in cross-section, are driven on or near the footwall, in the vein. Mounted IR-S 49 Jackhamers and 40 per cent Hercules dynamite are used for drilling and blasting. Stope drilling is done with IR-S 49 mounted Jackhamers or L 74 drifters, using steel up to 7 ft. 8 in. in length. In the glory hole, IR-S 39 and S 49 Jackhamers are used exclusively. Blasting is done in the stopes and glory hole with 30 or 40 per cent dynamite, depending on the nature of the ground.

Chute centers are spaced at 25 ft. All chutes are equipped with standard two-gate iron doors. From the loading chutes and drifts, broken ore is trammed by hand to the shaft and dumped into the 70-ton skip-loading pocket on the 160-ft. level. Compressed air for underground operations is furnished by an 800-cu.ft. C.P. air compressor. Mine water is pumped to the surface by an automatically controlled 200-g.p.m. Sterling deep-water pump. Power is obtained at 440 volts from the Pacific Gas & Electric Company. Water for plant and camp use is furnished by the Georgetown Divide Water Company.

Headframe and steel surge bin

From the loading pocket the ore, broken to 10 in., is hoisted to the surface and dumped automatically into the steel surge bin attached to the headframe, whence a 24-in. incline belt conveyor delivers it to the 500-ton wooden mill bin, as shown in Fig. 1. Through two hand-controlled gates, the ore is then fed periodically to a duplex Hadsel mill for crushing. The mill consists of a short length of cylinder, provided with buckets on the inner periphery; breaker plates of chrome steel, independent of the cylinder, on which the contents of the buckets discharge; and a concrete classifier-tank base. Motor drive is by double-reduction spur gears, motor and gearing being between the two overhung units. The motor is 100 hp. Mill diameter is 24 ft.; speed, 2.66 r.p.m.; buckets, 3 ft. square by 2 ft. deep, are 3 ft. apart. The bucket lips are of manganese steel, and the wheel proper is of steel plate. Crushing and grinding is effected by impact and attrition as described on page 229 of the April, 1932, issue of *Engineering and Mining Journal*.

At present only one unit is in operation, crushing from 150 to 215 tons of material a day. Power consumption is 60 to 70 hp. Total capacity of the mill varies up to 500 tons. Steel consumption per ton has not been determined as yet, but that indicated is low. Classification takes place inside the mill, and is effected by the rising current. The mill overflow or finished product, containing from 15 to 23 per cent solids, leaves the mill-tank base, as shown in Fig. 2. The length of the aperture is 7 ft., and constant, whereas the width is variable. By means of geared tracks, the width can be increased from 1 to 12 in. Obviously, a reduction in width means a smaller discharge area, higher discharge velocity, and coarser grinding. Another factor to influence grinding is the amount of water added. A screen analysis of the mill discharge follows: plus 80 mesh, 8 per cent; minus 80 plus 100 mesh, 6.5 per cent; minus 100 plus 200 mesh, 21.0 per cent; minus 200 mesh, 64.5 per cent.

From the Hadsel mill the ground ore flows by gravity to a Link-Belt drag classifier. Its length is 41 ft.; the speed, 15 ft. per second; and the motor, 5 hp. Classifier slime, which constitutes 55 per cent of the feed, flows to a six-cell Kraut flotation machine operating in

Details of Operating Costs

	Labor	Supplies	Water Power	Miscellaneous	Company Insurance	Total	Cost per Ton
Hoisting	$577.31	$10.00	$59.44	.....	$87.13	$733.88	$0.153
Development	552.71	312.00	59.43	.....	83.35	1,007.49	0.210
Glory-hole mining	428.53	401.00	59.44	.....	64.88	953.85	0.200
Grinding	559.50	10.00	343.87	$10.00	9.91	933.28	0.195
Treatment	562.65	250.97	284.44	32.55	9.97	1,140.58	0.239
Bullion expense	.....	.....	.....	41.69	.....	41.69	0.009
General expense	.....	.....	.....	99.88	.....	99.88	0.021
Plant expense	22.50	.....	.....	.....	.....	22.50	0.005
Totals	$2,703.20	$983.97	$806.62	$184.12	$255.24	$4,933.15	$1.032

Number of men employed, 24; total man-shifts, 661; daily tonnage, 178.

closed circuit with the classifier sand distributor. In other words, the rough flotation concentrate is collected in a small sump and pumped back continuously to the classifier sand distributor by a Krogh sand pump. The classifier slime, or flotation feed, contains from 6 to 10 per cent solids, and a screen analysis shows 7 per cent on 200 mesh, and 93 per cent through 200 mesh. Reagents used are: Aerofloat No. 15 and potassium amylxanthate. Consumption is 0.006 lb. and 0.06 lb. per ton, respectively. Classifier sands flow by gravity to four 8x30-ft. leaching tanks. Barren solution is used to facilitate the flow of the classifier sands to the leaching tanks. Overflow of leaching tanks returns to an 8x30-ft. circulating tank. Leaching tank effluent, or pregnant solution, is collected in an 8x30-ft. steel tank, whence an IR-V 3 centrifugal pump delivers it to the precipitation plant. The circulating solution in the storage tank is pumped back to the leaching tanks by an IR-V 3 centrifugal pump, as it builds up in value. Leaching operations require about nine days. Consumption of cyanide is 0.16 lb. per ton, and that of lime 0.5 lb. per ton. Screen analysis of

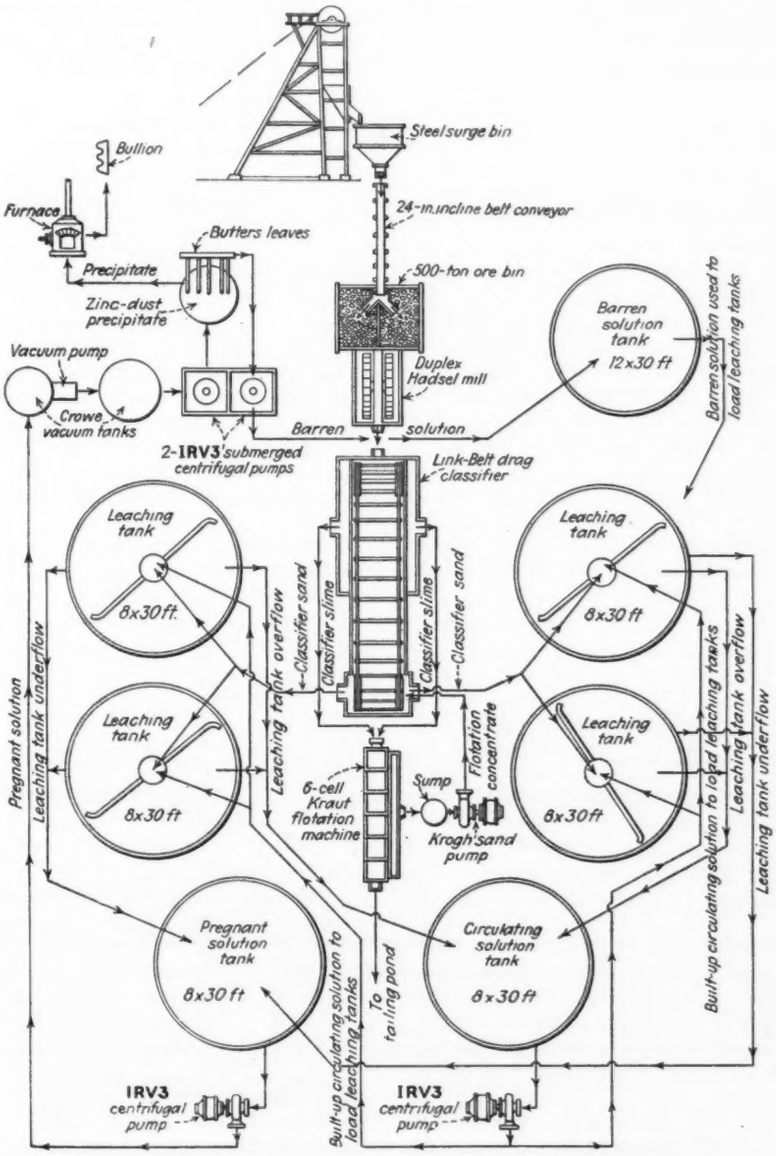
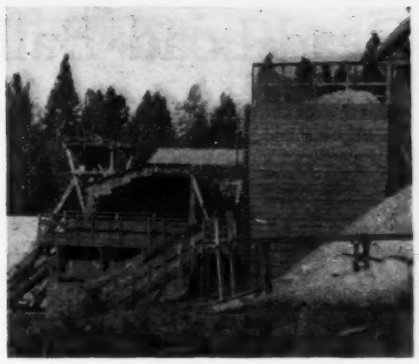


Fig. 1—Metallurgical flowsheet of the Beebe Gold Mining Company



Ore bin and Hadsel mill

the classifier sands indicates 73 per cent on 200 mesh, and 27 per cent through 200 mesh.

Flotation of classifier slime gives a tailing assaying about 10c. per ton. Application of flotation also permits finer grinding, with a consequent greater slime overflow. Coarser grinding, on the other hand, allows better and faster leaching; in fact, recoveries up to 96 per cent have been obtained in the leaching tanks by coarser grinding. However, with flotation, the total recovery is over 90 per cent.

At the precipitation plant the pregnant solution is passed successively through Crowe vacuum tanks and a zinc-dust precipitator containing several 3x3-ft. Butters leaves. Zinc dust is added at the rate of 0.03 lb. per ton of

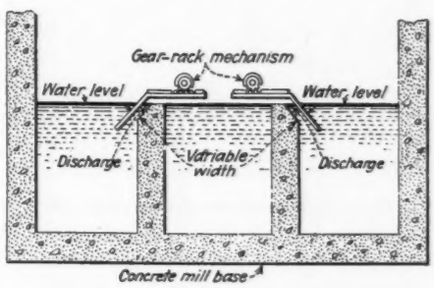


Fig. 2—Tank base of the Hadsel mill

solution. A vacuum pump and two submerged IR-V 3 centrifugal pumps are available to circulate the pregnant solution through the various apparatus, and to return barren solution to the 12x30-ft. storage tank. Ultimately, the precipitate is scraped off the Butters leaves, dried, and melted into bullion in a locally made, oil-fired melting furnace. The furnace contains a clay-lined pot of 125-lb. capacity, and a DFC oil burner using high-pressure air. The flux used contains the following ingredients: fluorspar, 5 per cent; niter, 12 to 15 per cent; borax glass, 5 per cent; and silica 5 per cent. Niter is mixed intimately with the precipitate, and the other ingredients are used as a cover. Operating costs are shown herewith. A material reduction in cost is expected to be made as soon as the mill can be operated at full capacity.

Dead-Load Ball-Mill Power Consumption

Alexander M. Gow

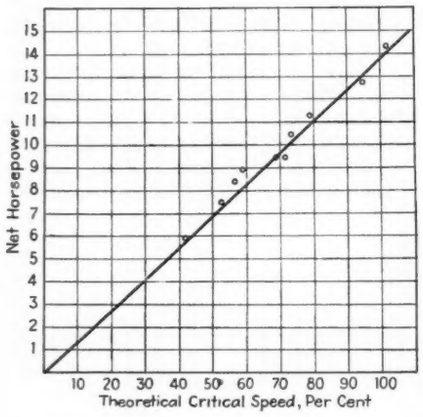
and

Morris Guggenheim

Assistant Metallurgist and Junior Metallurgist, respectively, U. S. Bureau of Mines, Mississippi Valley Experiment Station, Rolla, Mo.

ONLY a part of the power required to drive a ball mill is used in imparting grinding action to the balls; some is expended in frictional losses outside the mill. This part of the motor output, called "dead load," is approximately equal to the power that would be necessary to rotate a flywheel of the same weight and of similar design. The analogy has facilitated determination of dead load in the power balance sheet.

In the laboratory investigation of ball milling being conducted at the Mississippi Valley Experiment Station of the U. S. Bureau of Mines in cooperation with the Missouri School of Mines and Metallurgy, at Rolla, Mo., determination of dead load and subsequent calculation of the net power chargeable against grinding have become a matter of routine; but only recently has a dead-load test been made on a commercial ball mill—a 6x4 ft. unit in the lead belt of southeast Missouri.



Dead-load curve for 6x4 ft. ball mill

The mill, motor driven through belt and spur gear, has a manhole in the breast. It was emptied, and 2-in. plank bulkheads were built into each end, so that the inside length was reduced from 4 to 2 ft. The mill was turned so that the manhole was at the top, and then the ball charge was packed into the shortened mill to fill it completely. With the manhole cover replaced, the mill became a flywheel whose weight equaled the dead weight of the operating ball mill one-half full of balls.

The regular alternating-current motor was replaced by a variable-speed, direct-current motor, with speed controls and meters necessary for determining the motor input and calculating motor output. By recording the power output at various speeds, dead-load data were obtained. The accompanying curve gives, at different speeds, this net horsepower necessary to overcome external friction. The figures might be increased on account of the unbalanced load on the trunnions and because the countershaft and gears transmit the greater amount of power used in normal operation. The determination, however, gives a figure that represents minimum frictional loss outside the mill, and therefore should not be charged against grinding. The dead-load curve, a straight line over the range investigated, and which if extrapolated linearly passes through the origin, is of the same type as that of the laboratory mill of smaller dimensions and was forecast by the laboratory testing.

Ordinarily the mill runs at 80 per cent of the theoretical critical speed, and the gross power input to the motor is 87 hp., as determined by a polyphase recording wattmeter. The motor efficiency is 90 per cent (by factory efficiency curves), and the dead load, as may be seen from the appended curve, is 11 hp. This leaves 67.3 hp. to impart grinding action to the balls and to represent the energy spent within the mill. The dead load, therefore, under the operating conditions of this mill, amounts to 13 per cent of the gross power input; and, with the 10 per cent motor loss, the total loss becomes 23 per cent of the power input to the motor. Only 77 per cent of the power required to operate this mill is expended inside the liners.

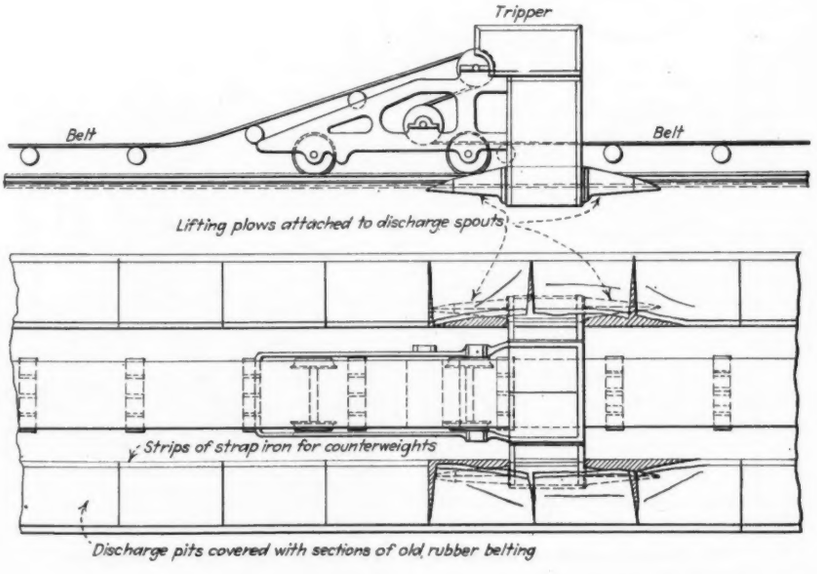
Published by permission of the Director, U. S. Bureau of Mines.

■ ■ ■

Automatic Sealing of Ore Bins

AN INNOVATION at the crushing plant of the New Cornelia mines, Phelps Dodge Corporation, at Ajo, Ariz., is the method of avoiding the escape of dust while the storage bins are being charged by the moving belt-conveyor tripper. This is accomplished by the arrangement shown in the accompanying sketch. The two channel-like openings at the top of the bin are covered with oblong strips of old rubber belting—an ideal material for this purpose because of its flexible nature and light weight, offering little resistance to the discharge spouts of the tripper. To the loose end of each rubber section, or flap, is bolted a piece of strap iron, as a counterweight, to insure proper seating of the cover. Raising and lowering the cover flaps by the moving tripper is effected by two triangular plow-like lifters, made of $\frac{1}{8}$-in. plate, secured to the tripper discharge ends.

Application has been made for a patent covering the device described.


Tripper
Belt
Belt
Lifting plows attached to discharge spouts
Strips of strap iron for counterweights
Discharge pits covered with sections of old rubber belting

The Rôle of Mining in the Development of Civilization

MAN AND METALS. *By T. A. Rickard. New York, N. Y.: Whittlesey House, McGraw-Hill Book Company. Two volumes; 1,068 pp.; illus.; $10.*

Reviewed by Dr. T. T. Read

IT IS a pleasure to be asked to review Mr. Rickard's epoch-making work, since it happens to appear almost simultaneously with my own much smaller one that has for its theme the relation between minerals and civilization in the past hundred years. Mr. Rickard takes the whole span of human history for his field, and while the title might lead one to suppose that he discusses only metals, the first chapter, "The Ages Without Metal," necessarily concerns itself with stone and clay, while Chapter XIV treats of "The Use of Coal in the World's Workshop." It is interesting to learn that the compelling impulse to undertake the immense labor involved in such a work came from the publication, by his fellow-student under Huxley, H. G. Wells, of his widely read "The Outline of History." Mr. Rickard thought (and the rest of us will agree) that Wells failed to "pay proper regard to the part that mining has played in the development of civilization." Having the qualifications for, and being willing to undertake the labor of, writing a work in which the rightful place of minerals along "man's rough road" is set forth, the rest of us should be eternally grateful to him for doing it.

Earlier material on the history of minerals fell mostly into one of two categories. Either it was written by archeologists or sociologists who were so unversed in mineral technology as, for example, to be unable to clearly differentiate copper, bronze, and brass, and to distinguish between melting and smelting, or else it was the result of researches by men who, because they thoroughly understood mineral technology, confined themselves to its history and ignored its relation to the maintaining and improving of the conditions under which men live and work. The right result can only come through a combination of these two viewpoints, a sort of stereoscopic view of the subject, and this Mr. Rickard has, for the first time, provided. No one in the mineral industry will need to be informed that he tells the story in the lively and readable style so characteristic of him, or that he brought to bear on the task a wide knowledge not only of mineral technology but of general culture.

The labor involved in preparing such a book is justified because the finished product serves so many useful purposes. It makes available to the general public that which every really educated man ought to know, but which has not been easily available. It should be required reading in every university course in which students are expected to grasp the salient facts in the history of civilization. It makes available to archeologists, sociologists, and historians a work of enlightenment and reference that will enable them to avoid the persistence of errors that have previously been made while endeavoring to contribute to a more precise knowledge of earlier times. To the men in the mineral industry it brings a broadening of their intellectual horizon through instilling a realization that in producing mineral they were not simply making a living for themselves, but were also making a notable contribution to human progress.

Finally, it will be an inspiration to mineral technologists to devote themselves to the task of clearing up the many points on which no final decision can be reached until all the evidence is in. Thus, on p. 155, Mr. Rickard says, "The production of brass means the mixing of zinc and copper in definite proportions, with deliberate intent and with knowledge of the metallurgic art. This could not be done until the metal zinc was discovered, at the end of the Medieval period—in 1509." Lathrop, on the other hand ("History of the Brass Industry," p. 40), asserts that all brass made in Europe till 1781 (and in America till 1802) was produced by "cementing" copper with calamine. P. Neogi claims that zinc was produced as a separate metal in India as early as the seventh century and H. T. Chang argues that it was similarly produced in China at least as early as the eleventh century. Since Mr. Chang is able to cite from the literature of that period statements that coins were then made from eight parts of copper, four parts of "black tin" (lead), and two parts of "white tin" (zinc), and actual analysis of the coin reveals that they do contain 55 per cent copper, 26 per cent lead, and 13 per cent zinc, the evidence seems pretty strong that as early as the eleventh century the Chinese not only produced zinc as a separate metal, but made their coin metal by mixing copper, lead, and zinc. It will take time and much study to finally solve questions such as these, and it is to be hoped that Mr. Rickard's example will stimulate more mineral technologists to undertake historical research.

In conclusion, it should be added that the publishers deserve at least some of the praise for the issuing of so notable a contribution to literature. It must require no little courage to bring out a two-volume work in the present state of the book market, and it deserves to be rewarded by a good sale.

■ ■ ■

PETROGRAPHY AND PETROLOGY. *By Frank L. Grout, Pp. 522; illus. New York, N. Y.: McGraw-Hill Book Company, Inc.; $5.*

The subject matter of this excellent textbook is divided into eight parts. Part I, the introduction, contains brief lists of criteria useful in distinguishing igneous, sedimentary, and metamorphic rocks, a list of the commoner rock-forming minerals and their compositions, concise directions for handling loose mineral grains and rock powders, and samples of rock descriptions used in making engineering, economic, and petrographic reports.

In Parts II and III Professor Grout makes proper distinction between the *petrography* of igneous rocks, Part II, and the *petrology* of igneous rocks, Part III. The petrographic section treats of forms, structures, textures, the mineralogy, order of crystallization, classification, and nomenclature of igneous rocks, which are divided into "clans" for the purpose of discussion. The discussion is enhanced by good illustrations.

The petrologic section treats of the mode of formation of igneous rocks, especially in relation to earth processes. This section, which is highly theoretical, contains an excellent and impartial presentation of the ideas of other authors with respect to the physico-chemical processes and conditions related to magmas, and their emplacement and behavior. Numerous diagrams and other illustrations aid in clarifying the discussion. Brief lists of criteria helpful in interpreting certain phases of petrogenesis add interest to the section.

The same treatment is accorded both the sedimentary rocks, Parts IV and V, and the metamorphic rocks, Parts VI and VII, although must less space is devoted to them than to the igneous rocks.

Part VIII consists of various sorts of tables which should prove helpful to the student; and in addition there is an extensive bibliography of selected

reading which is a valuable feature of the book.

The work is so excellent and so commendable that one hesitates to criticise any part of it adversely. Perhaps the sections on the sediments might have been somewhat improved by a more extended discussion of the correlation of beds by means of heavy mineral residues, and possibly the sections dealing with the metamorphic rocks might have been made a little more complete by the inclusion of a discussion of retrogressive metamorphism, phyllonitization and diaphthoresis, conceptions so important in the interpretation of past changes in geologic environment. These are but minor omissions, however, that do not reflect adversely on the general good quality of the book.

This volume should be stimulating not only to students, but to all who may be interested in petrography and petrology. The presswork and typography are excellent, and both author and publishers are to be congratulated on the production of such a fine piece of work. R. J. COLONY.

MINERAL INDUSTRY DURING 1931. VOLUME XL. *Edited by G. A. Roush. New York: McGraw-Hill Book Company; pp. 735; $12.*

This hardy annual, covering the statistics, technology, and trade of the mineral industry, is so well known by this time that little can be said that has not already been said. It has long since become regarded as a standard reference and, each year, the editorial offices of *Engineering and Mining Journal* are besieged weeks in advance of its date of publication with inquiries as to when it will appear. The form in which the data are presented is more or less standardized; only the substance changes and sometimes the contributors. In the latter respect, one notes that only one new name appears this year, that of S. D. Strauss, who has written the article on zinc, exclusive of the metal's metallurgy, which article was prepared by an unnamed author last year. W. R. Ingalls contributes the pages on the metallurgy of zinc as usual. Otherwise the list of contributors is unchanged, aside from the fact that three chapters signed last year appear anonymously in the new volume. The regretted death of Dr. George Frederick Kunz made it necessary for the editor, Dr. Roush, to complete the work of preparing the two chapters customarily contributed by Dr. Kunz and on which he had already done considerable work when he laid down his pen.

GEOLOGIA PRACTICA. *By Emilio Cortese. Milan, Italy: Ulrico Hoepli. Pp. xvi, 447; price, lire 20.*

A pocket volume, in flexible binding, containing information on the mineral deposits of Italy, building and other construction material. It has been compiled to be of especial value to the Italian mining engineer and geologist.

INTRODUCTION TO METALLOGRAPHY. *By L. W. Eastwood. Houghton, Mich.: Michigan College of Mining and Technology; 10¾x8½ in., zincograph; $2.50.*

This text discusses the general nature of metals and alloys, their constitution, and their reaction to heat-treatment and mechanical work. Illustrations comprise 65 figures and 22 photomicrographs. Two appendices provide detailed information on the use of photographic methods in the metallographical laboratory and on the preparation of polished specimens.

OUR MINERAL CIVILIZATION. *By Thomas T. Read. Baltimore, Md.: Williams & Wilkins Company; pp. 165; $1.*

As one of a so-called "Century of Progress" series, under preparation by the publisher, this little volume admirably fulfills its purpose by demonstrating the part metals and minerals have played in man's attainment of his present civilization. The treatment is popular, as is proper, because thus the greatest number of readers will be served. Nevertheless, many who are engaged in one or another phase of the mineral industry will find the book an aid to memory where the uses of minerals are concerned. Thanks to a not-too-obtrusive humor and a philosophical bent of mind, Dr. Read has turned out a book that is interesting as well as informative.

CINNABAR IN SOUTHWESTERN ARKANSAS. *George C. Branner, Information Circular 2, Arkansas Geological Survey, Little Rock, Ark., Pp. 51, illustrated.*

This summary of available information on the most recent and at the outset sensational discovery of quicksilver in Arkansas is timely. Proof is lacking as to any disclosure of the metal prior to 1930, when the find referred to was made. The discovery area, as now defined, on the Athens Plateau, in the southwest, lies mainly in Pike County, but extends also into Howard and Clark counties. At least 34 separate occurrences are known, most of them being nearly in a straight line about 20 miles long. Three types of deposit have been noted, the principal one being that of fracture fillings, in steeply dipping Mississippian sandstones; and the others dissemination in sandstone and in association with vein quartz, respectively.

Development of cinnabar in the area had been undertaken by five companies and one individual up to May 28 last. Of these, the Arkansas Quicksilver Company, Prescott, Ark., installed a two-tube, Type D Gould retort, and obtained from 65,000 lb. of rock an average yield of 1.99 per cent. At the end of May it was retorting about 3,000 lb. of ore per 24 hours, the daily yield being 40 to 60 lb. of metal. Twenty-five flasks sent out on April 26, 1932, comprised the first shipment of Arkansas quicksilver in bulk. The "C" Mining Company, Murfreesboro, Ark., erected a crusher and a retort, rated at 10 tons, and a condenser, and ran a short time. Southwestern Quicksilver Company, Hot Springs, Ark., installed an experimental three-tube, Type D retort, and treated 39,820 lb. of ore in November, 1931, for an average yield of 1.73 per cent, the range being from 0.25 to 2.84 per cent. In April, last, the company placed in commission a crusher, a rotary, oil-burning Gould kiln, of 15 to 20 tons' daily capacity, an eight-tube condenser and a dust separator. Results have not been announced.

As for commercial possibilities, the bulletin states that, with the knowledge available, no clear idea of the values to be expected in the area, as now defined, can be expected. As for the depth to which deposition may continue, no dependable estimate can be made. At present, the bulletin says, the situation is encouraging for systematic careful prospecting. The report presumes that, generally speaking, mining will have to be conducted on much the same basis as in Western United States—that is, low-grade orebodies will have to be located and their average quicksilver content estimated, followed by such exploration as the price of quicksilver may justify. The bulletin then makes suggestions as to prospecting and development, concluding with a tabulation of 34 occurrences in place, giving in each instance the location, names of fee owner and lessee, the formation, structure, prospecting work performed, and, briefly, the nature of the ore occurrence. Several maps facilitate the understanding of the geography, physiography, and geology of the occurrences.

A. H. HUBBELL.

SURVEY OF MINES: CANADA AND NEWFOUNDLAND. *Toronto, Canada: Financial Post; 183 pp.*

This is the seventh annual edition of a handbook that deals with almost every mine of importance in Canada and Newfoundland. New fields have been given adequate attention, and new companies are listed. A list of mining engineers, mine managers, and geologists is provided, with their affiliations. Statistics supply condensed information condensed on mineral production by provinces.

KEYSTONE METAL QUARRY CATALOG, 1931. *Eleventh edition. New York: McGraw-Hill Catalog and Directory Co.; pp. 342; $15.*

This standard volume contains a completely indexed collection of manufacturers' catalogs of machinery, supplies and equipment.

Among Contributors To This Issue

Donald M. Fraser graduated from the University of Oregon in 1925, having also attended the Oregon State College and Stanford University. He has served as assistant instructor, or assistant professor, at the University of Oregon, Occidental College, Hunter College, and Dana College. A university fellowship at Columbia, 1928-29, was followed by professional work, and mapping for the California State Mining Bureau, and he was awarded the degree of Doctor of Philosophy, Columbia University, in 1932. He is now instructor in geology at Lehigh University.

T. T. Read (Columbia, 1902) served on the faculty of the University of Wyoming soon after graduation, becoming associated with North American Copper and Oxford Copper subsequently, earning the degree of Ph.D. at Columbia meanwhile. In 1907 he was appointed professor of mining and metallurgy at Colorado College, from which he resigned soon after to become professor of metallurgy at the Imperial Pei-Yang University, Tientsin, China. From 1910-15 he was associate editor of *Mining & Scientific Press;* from 1916-18, metallurgist with New Jersey Zinc; 1919-26, chief of information service division of the U. S. Bureau of Mines; 1926-28, assistant secretary, A.I.M.E. In 1928 Doctor Read was appointed Vinton professor of mining engineering at the School of Mines, Columbia University.

G. E. Lynch (Massachusetts Institute of Technology) had an unusually varied professional experience in the design and construction of mechanical equipment before engaging in plant design and process development, which eventually led to specialization in the problems of dust control. Since 1913, when he established as an independent consulting engineer, with headquarters in Los Angeles, Calif., he has become an authority on the subject; and to him credit is largely due for the successful operation of dust-control equipment adopted by progressive companies throughout the Southwestern United States in particular, and, as the article in this issue indicates, elsewhere as well.

✦

F. P. Brunel, since graduation from the Colorado School of Mines in 1912, has worked—as miner, engineer, chemist, and foreman—in the engineering and mining departments of Calumet & Arizona, New Cornelia, and Shattuck Denn companies in the Southwest. He was connected with the development of Ajo Consolidated, and was recently in charge of the work of concreting the Denn shaft, at Bisbee, which he describes elsewhere in this issue.

✦

John B. Huttl is a graduate of the M.A.N. Technical School, and Stadtische Bauschule, Nurnberg, Germany. He has had operating experience in the Ruhr district, Germany; in Spain, Mexico, and Central America. During 1929-31 he represented *Engineering and Mining Journal* as assistant editor in the Southwest.

C. V. Avetisian is associate professor of metallurgy at the Moscow Institute of Non-Ferrous Metals, and research metallurgist at the Moscow State Research Institute of Non-Ferrous Metals. He was graduated from the Moscow Mining Academy in 1928, and served subsequently as a member of the operating staff of the Zangezur copper smelter, in the Caucasus.

Charles A. Porter, who began his professional career as assistant surveyor with Consolidated Mercur Mines, in Utah, in 1904, draws on an experience rich in technical scope, covering prospecting and leasing, sampling and assaying, chemical and metallurgical control, and mine management in many Western states. Ideas he has advanced in article form have been born of experience in the field, usually the result of effort to clarify thought and to utilize geology to the fullest practical value.

Rudolf Gahl (Universities of Berlin and Goettingen) has specialized in research in chemistry, electrochemistry, and hydrometallurgy, with Detroit Copper, Phelps Dodge, and other Western copper interests; and with Cerro de Pasco, in Peru. During 1914-18 he was metallurgist to Inspiration Consolidated. He has contributed much, in process and in print, to technical advance.

R. J. Colony is assistant professor of geology at Columbia University, a graduate of that institution and of Cooper Union. Subsequent to professional work in Mexico and the United States he was appointed, in 1900, to the faculty of Cooper Union, and to his present position at Columbia in 1922.

W. V. DeCamp, of Jerome, Ariz., has resigned as general manager of United Verde Copper.

Andrew W. Newberry announces removal of his office to 911 Equitable Building, Denver, Colo.

Enoch Perkins, manager of mines for Mutual Chemical of America, is expected in New York, from New Caledonia, on Jan. 12.

Guy Riddell returned to New York late in November, after completion of a two-year contract with the Soviet government that involved general consulting work in the rare-metal and non-ferrous metal industries.

A. P. Selby-Davidson, director of the firm of Garrett & Davidson, metallurgists, of Sydney, N.S.W., was recently in New York. After an extensive survey of plants in North America, he plans to return to Australia via San Francisco.

Dr. Chester W. Washburne, who recently lectured at the Montana School of Mines on the subject of structural dikes, is expected to return in the spring to deliver another series, on faulting.

M. Barzin, formerly general manager of Union Minière, has succeeded Count Marc Minette d'Oulhaye as managing director of Geomines, operating in Katanga.

Noel Griffin, of Rhodesia, is making an inspection of the Wiluna gold mines.

Anson H. Smith on Nov. 5 celebrated the fiftieth anniversary of the founding of the *Mohave County Miner,* of which he has been the editor since 1882, when the first edition was issued in the now abandoned town of Mineral Park, Arizona Territory.

C. H. Aldrich, technical manager of United States Metals Refining, sailed on Nov. 18 for England, where he is supervising the new fire refinery he designed for British Copper Refineries, at Prescot, Lancashire. This plant, owned jointly by American Metal and British Insulated Cables, will soon be operating.

E. H. Greig, underground superintendent of Burma Corporation, is in Australia.

Ivor E. Hughes, formerly with International Nickel at Copper Cliff, Ont., has joined the research staff of Delta Mines Syndicate, Saulte Ste. Marie, Ont.

G. H. Hutton and **A. D. Storke,** of London, were recently elected members of the Institution of Mining & Metallurgy. The name of **A. G. Macdonald,** of the Federated Malay States, was transferred from associateship to membership.

Hon. R. Randolph Bruce, who for many years owned and operated the Paradise mine, in the Windermere district of British Columbia, and recently completed a term as lieutenant-governor of the Province, was married in Montreal on Oct. 18 to Edith Badgley Molson.

J. C. Henderson, after seventeen years of association with the Driver Harris Company, of Harrison, N. J., has resigned to devote his entire attention to heat-resisting castings, particularly those of the nickel-chromium-iron group, in the development of which he pioneered in 1915.

J. V. N. Dorr has been elected to honorary membership of the Chemical, Metallurgical & Mining Society of South Africa, and appointed a member of the senate of the department of Chemical Engineering, Columbia University, New York, and an advisory member of the College of Engineering, New York University.

F. S. McNicholas draws attention to misinterpretations that occurred in the preparation of his article in the October issue, on "Blasting Big Rounds Electrically." Contrary to an impression that might be inferred in the introductory paragraph, the technique refers to practice in blasting up to several thousand holes simultaneously. Tables I and II, with data and calculations, refer to one blast only, and the figures in the penultimate paragraph were used to form the basis of general conclusions, not for supplying the data in Table II, as stated.

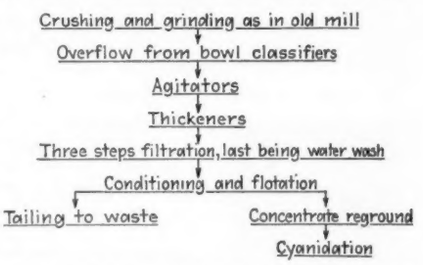
P. R. Bradley, Jr., sends details of a novel portable compressor plant, mounted on an old Tonopah fire truck, in use on the lease held by Sheriff W. H. Thomas and the Booth Brothers, of Tonapah, at the new gold camp of Clarkdale, Nye County, Nev. Power is furnished from the truck motor, and the take-off is the drive shaft of the truck. When the unit is to be moved, the chain is returned to the sprocket on the rear axle. The compressor used is a 4¾x4¾ Clayton machine, supplying air for one Jackhamer. A shaft round is drilled at an expense involved in the cost of 4 to 5 gal. of gasoline. The plant proper consists of the compressor, a centrifugal water pump, a water tank of a size sufficient to insure proper cooling, and an air receiver. The truck carries the drilling equipment and blacksmith tools. Hand-sharpened Carr bits are used.

Metallurgical Trends in Canada

According to information received from Maurice W. Summerhayes, general manager, the new flowsheet for 1933, to be adopted at the plant of Wright-Hargreaves Mines, Ltd., at Kirkland Lake, Ont., on a basis of 800 tons per day, will involve the addition of one Hardinge spiral classifier and a 5-ft. Andrews (kinetic-elutriator) classifier. These are being employed as test units on the bowl-classifier floor for the purpose of relieving the three Dorr bowls, which are working to over-capacity, necessitating the maintenance of an overflow of 9 : 1, solution to solids, to get 95 to 98 per cent minus 200 mesh. It is expected to reduce this to a ratio of 5½ : 1 with these extra units.

The next step will be to replace three 34-ft. Dorr thickeners with one 18-ft. diameter Genter, with a capacity or filtering area of 2,300 sq.ft., to thicken 600 tons per day; the remaining Dorrs receive 200 tons per day, as surge tanks, and to permit a direct comparison with the Genter, receiving pulp from the primary agitators. The thickened discharge (50 per cent solids) will go to one pair of Oliver filters; cake, carrying about 20 per cent moisture, to repulper, each at 1 : 1, thence to a second pair of Oliver filters, each of the foregoing to have barren-solution sprays. The third pair of Oliver filters will receive repulped material that will be sprayed with fresh water, to wash as much lime and cyanide from the pulp as possible, in anticipation of the next step. The cake from the third filtration will be brought to a ratio of 3.54 : 1 density, with fresh water, further reducing cyanide and lime content. This pulp will be preconditioned in one main conditioner, thence going to a smaller conditioner at the head of each bank of three sets of flotation cells, consisting of two banks, of six cells each, of the latest type 56-in. Fagergren cells and one bank of six cells No. 24, Sub A Fahrenwald. In the case of the former, the middlings will go to two Fagergren cleaners; but in the Sub A's the conditioned pulp will feed to No. 2 cell for roughing; and by launders all middlings from Nos. 3, 4, 5, and 6 cells will be returned to No. 1 for cleaning. As an alternative in the case of the Fagergren cleaners, a pair of standard Kraut cells may be used. On the same floor as these three banks will be a 6-cell bank of Sub A's, No. 12, on which all pilot-mill tests were made. These will serve as a guide unit for reagent control. The flotation concentrates will pass to a small steady tank, thence to a 60 sq.ft. Genter thickener, previously used in pilot-plant operations.

Automatic samplers will be installed at all points in the circuit, and a Haultain type of density recording apparatus at the conditioners. From the 60-ft. Genter the thickened concentrates will, at 1 : 1, pass to a 5 ft. × 22 in. silex-lined Hardinge mill, for regrind with pebbles, where quicklime will be added. Discharge from ball mill will be diluted with barren solution and brought up to 4 : 1 ratio, and the necessary cyanide strength, and delivered to any one of a series of super agitators, arranged for a time agitation of up to 96 hours. At the end of the agitation period the pulp will be settled, and supernatant pregnant solution decanted and passed through a Butters or an Oliver-Borden type of vacuum clarifier; the clarified preg-



nant solution will go to a precipitation press operating independently from precipitation of the main mill circuit. The settled pulp will be reagitated with water, settled and then repulped with water and refiltered, to remove dissolved gold.

The cyanide plant for the treatment of the flotation concentrate is designed for a future contemplated mill capacity of 1,000 tons per diem. The flowsheet, as above outlined, will serve as a guide under working conditions on an 800-ton basis; and, assuming that the Genter and other units prove efficient, a second Genter will be installed, further reducing mill floor area, and at the same time increasing mill capacity.

At present, the ball-mill feed is all through ⅜ mesh, but it is probable that, with a capacity of 1,000 tons in 1934, larger grinding units would be installed, and probably a large set of rolls to reduce the feed all to minus ⅝ in.—to go direct to the classifier, overflow to tubes, coarse return passing through the ball mills.

Portable compressor plant at lease of Booth Brothers, Clarkdale, Nev. Tom Clark, discoverer and owner of property, on right. The Booth Brothers are alongside the unit described by P. R. Bradley, Jr., elsewhere on this page.

Sir Auckland Geddes, formerly British ambassador to the United States, now chairman of Rhokana Corporation (Rhodesia) and of Rio Tinto (Spain), arrived in New York late in November to attend the international conference of copper producers, meeting to discuss and to allocate production quotas and to consider other business of importance to the industry. Other representatives from countries foreign to the United States included **James Y. Murdock,** president of Noranda Mines (Canada); **R. H. Channing, Jr.,** president of Hudson Bay Mining & Smelting (Canada); **Robert A. Stanley,** president of International Nickel (Canada); **S. S. Taylor,** managing director of Rhokana Corporation (Rhodesia); **Camille Gutt, Felicien Cattier,** and **Ferdinand Pisart,** representing Union Minière du Haut-Katanga (Belgian Congo). Representatives of domestic companies included **Cornelius F. Kelley,** president of Anaconda Copper; **Dr. Otto Sussman,** president of American Metals and representing Roan Antelope; **Stephen Birch,** president of Kennecott Copper, and **Louis S. Cates,** president of Phelps Dodge Corporation.

Alfred D. Flynn, director of Engineering Foundation, announces the publication of a pamphlet prepared by the Education Research Committee of that organization in the form of a message to young men, parents, and teachers, describing the profession of engineering. Copies are obtainable through the Foundation, at 29 West 39th St., New York City.

Frank N. Thomas, a member of the technical staff of Broken Hill South, Broken Hill, Australia, has been appointed underground manager of the South mine, succeeding **H. H. Carroll,** who recently became assistant general manager of Wiluna Gold, Western Australia.

Carl Hedman, general superintendent on the Mesabi Range for the M. A. Hanna company, has resigned. His place has been taken by **W. E. Craig,** formerly superintendent of the Susquehanna mine.

Adolph Kuechler, of Ohio State University and Missouri School of Mines, has been appointed instructor in ceramics at the Montana School of Mines.

D. D. Wilson, formerly works manager of the Detroit plant of U. S. Gypsum, has been appointed manager of the company's East Chicago, Ind., plant.

E. J. Gourley has resigned as Inspector of Mines, Western Australia Mines Department, to become manager for Norseman Gold.

Dr. Anton Gray has returned to London from Serbia, Yugoslavia, where he went recently to make a geological investigation for Trepca Mines.

Louis J. Joubert has returned to California from New Guinea, after two years spent in placer engineering work in the interior.

R. C. Riley is returning this month to Nagpur, India, from England.

Obituary

Thomas Taylor, until recently general superintendent of the smelter of United Verde at Clarksdale, Ariz.; at Clarksdale, on Nov. 28; aged 67. Mr. Taylor was born in Swansea, Wales, emigrated to the United States at the age of fourteen, returned to Wales to specialize in smelting practice, and in 1884 became associated with the Clark interests in Montana. He took part in the Klondike gold rush in 1898; and, returning to the copper industry soon after, rose to a position of high responsibility on the staff of United Verde, contributing much to the technique of copper smelting in the United States.

A. J. Moore, until recently mining editor of the *Nevada State Journal;* at Reno, Nev., on Nov. 14; aged 71. Mr. Moore was born in Brockton, Mass., went to California in the 'eighties, and became Sunday editor of the San Francisco *Call.* After the catastrophe of 1906 he went to Goldfield, Nev., and had since been closely associated with mining work, acting as correspondent and contributor to *Engineering and Mining Journal.*

Amy Horne Jennings, widow of Sidney J. Jennings, vice-president of United States Smelting, Refining & Mining; at New York City on Nov. 8; aged 72. Mrs. Jennings was born in India, educated in England, and was married to Mr. Jennings at Kimberley, South Africa, in 1893. Two sons and two daughters survive.

Cuyler Adams, discoverer of the Cuyuna Iron Range; at Tryon, N. C., on Nov. 29; aged 80. Mr. Adams, who was born in Canton, Ill., went to Minnesota in 1870, and discovered the Range by magnetic observation. He was a member of the A.I.M.E. and the Rocky Mountain Club.

Philip L. Foster (McGill, 1882), formerly director of El Oro Mining & Railway, Exploration, Ltd., and Greene Cananea Copper; at New York on Nov. 15.

William E. C. Eustis, metallurgist and one of the founders of International Nickel; at Milton, Mass., on Oct. 29; aged 83.

James L. Belford, vice-chairman of safety and welfare department of American Smelting & Refining, and connected with that company for nearly twenty years; at New York, following an operation; aged 65.

David L. Summey, consulting engineer for United States Metals Refining; at New York on Nov. 14; aged 57. From 1905 to 1913 he was designing mechanical engineer, specializing in extrusion processes, on the staff of the Chase Companies. From 1913 he continued his work on extrusion with Scovill Manufacturing, and later became chief engineer of that company. Five years ago he was engaged as consultant by United States Metals Refining, to design an electric melting furnace as the first step in the development of a process for producing oxygen-free copper. At the Carteret, N. J., plant of this company the process he developed is now in successful commercial operation.

Hjalmar E. Skougor (Technical Institute of Copenhagen), designing and constructional engineer, contributor of the survey on silica in the review issues of *Engineering and Mining Journal* for many years; at Neshanic, N. J., on Nov. 7; aged 49. During Mr. Skougor's association with companies directed by Guggenheim Brothers he designed plants and camp sites for many important enterprises, including the plant of Chile Exploration, at Chuquicamata, Chile, comprising haulage system; milling, hydrometallurgical, and electrolytic plant; and buildings for a self-contained community. Similar work was undertaken for the Braden Copper, at Sewell, Chile. Evidences of his skill and ability are to be found also in the design of the townsite at the A. S. & R. property at Rosita, Mexico, and the city of Arvita, Canada, undertaken for the Aluminum Company of America. More recently Mr. Skougor was responsible for the design of the Maria Elena and Pedro de Valdivia nitrate plants, in Chile, for Anglo-Chilean Consolidated Nitrate, and as an independent consultant he contributed his experience to port, terminal, transportation, heating, and lighting projects in New York.

Cooperation Between Contributor and Editor

• *SINGLE-SPACED typewritten manuscripts are a nuisance to editor, compositor, and proofreader. Unless the writer of a contribution leaves enough white space for instructions to the mechanical department, particularly in the upper half of the first page, the manuscript, after preparation, must be retyped and checked against the original or sent to the compositor as marked by editor and copy reader. If a single-spaced article be sent direct to the compositor, he is compelled to decipher it as best he can. Work is slowed down and complaints develop. Time is lost and annoyance is inevitable.*

• *ALL WHO voluntarily assume the rôle of authorship will promote their own interests and facilitate the satisfactory handling of their manuscripts if they will observe a few simple rules, for which there is sound reason in the technical and routine mechanical requirements of publishing.*

• *A BACKING SHEET for use when typewriting, containing suggestions to authors, has been prepared. A copy of this will be sent on receipt of a request, addressed to the Editorial Department.*

Butane as a Fuel For the Mining and Metallurgical Industry

INDUSTRIAL

PROGRESS

Rudolf Gahl
Consulting Metallurgist
Berkeley, Calif.

RECENT DEVELOPMENT of huge quantities of natural gas has stimulated production of a number of interesting derivatives, among them propane and butane. What is being piped over long distances and sold as natural gas to many communities and industrial plants is essentially methane containing ethane and also some propane and butane. Methane, CH_4, being the principal constituent of natural gas, has essentially the same well known characteristics. Ethane, C_2H_6, the next heavier hydrocarbon of the series, is also a gas under ordinary conditions of temperature and pressure. Propane, C_3H_8, following ethane, is still a gas in the range of temperatures prevailing in our various climates, boiling around —40 deg. F. at atmospheric pressure, and can be liquefied by moderate pressures at ordinary temperatures. In consequence, it can be stored and shipped in steel flasks and tanks for use as a general-utility fuel and fuel for special purposes, as, for instance, metal cutting. Its separation is more difficult and expensive than that of hydrocarbons of greater molecular weight. For this reason as well as because shipping and storage containers must be strong to resist the pressures that develop in warm climates, transportation of propane over long distances for sale in competition with other fuels is economically impossible.

Butane, C_4H_{10}, the next in the series, is more favorable in this respect. It is liquid at temperatures somewhat below the freezing point of water at atmospheric pressure. When confined in flasks or tanks, it remains liquid up to much higher temperatures, these creating of course considerable pressures. However, the vapor pressures that are thus developed, can easily be coped with in designing containers. From the commercial point of view, butane may therefore be considered the natural gas fraction most suited for rail or boat transportation.

Individual hydrocarbons of a molecular weight higher than that of butane are not produced cheaply enough to be available as fuels.

The position of butane in the series of related hydrocarbons is responsible for some of its distinctive features. For instance, whereas methane requires about ten times its own volume for combustion, butane must have more than 30 (and gasoline and fuel oil still more). It so happens that 30 volumes is nearly the limit in the quantity of air that a can be drawn in by the gas jet of a burner, so butane accordingly is the highest hydrocarbon in the series that does not require air under pressure. A hydrocarbon of moderate molecular weight, moreover, may be the best where a flame of a certain length is wanted.

When burned as a source of heat, butane develops a heat of combustion of 102,000 B.t.u.'s per gallon (21,400 per pound). If it be considered with other fuels for use in applications where the fuel or combustion gases are not in contact with the material heated, as for heating a muffle, a crucible furnace, or a lead pot (for all of which butane has been used), the local cost of the heat units would probably be the most important consideration, although natural gas and butane might be preferred to coal or fuel oil because of greater ease of handling and controlling them and greater cleanliness. Where the combustion gases are in contact with the material to be heated, consideration of the chemical nature of the fuel will often shift the balance in favor of the pure hydrocarbon fuels. The fact that natural gas, propane, and butane are free from sulphur, unlike most coals and fuel oils, and that they burn without leaving a residue or depositing soot, are points in their favor. Instances of the use of butane under this classification include the operation of carburizing, annealing, and forging furnaces. So-called surface combustion systems have been found effective with butane, although impossible with very heavy fuels.

Although propane is being used almost exclusively where small quantities of a fuel gas have to be carried about in flasks, as for heating rivets or, in combination with oxygen, for cutting iron and steel, butane will also serve the purpose and should be used where it is kept on hand for power or other purposes, as in a mining camp. Its use in mine blacksmith shops in place of coke or charcoal saves time otherwise lost in heating the fuel bed to incandescence. In firing quicksilver furnaces, the soot that causes so much trouble when oil is used may be avoided by using butane instead. Butane, moreover, in furnace work in general, may be used to provide a reducing atmosphere that prevents oxidation. An assay furnace fired by it will appeal to the assayer not fortunate enough to have cheap electric current or gas at his disposal.

As a source of mechanical power butane is insufficiently appreciated. It may be used to advantage in combustion engines. The claim seems well founded that almost as much power can be obtained from a gallon of butane as from a gallon of gasoline, this being due to its having a slower flame rate and a higher octane number, which permits a higher compression ratio, which in turn results in better utilization of the available heat units. No atomization of the butane will be necessary, so that the mixing with air will be more easily and more thoroughly accomplished. Moreover, inasmuch as butane requires less air for combustion than gasoline, more of it can be fed for each cylinder charge, so that a given engine will be more powerful when run on butane.

Butane is finding a rapidly increasing field as a general utility gas. Shipping distance depends on freight rates, now the same as for refined oils and apparently high in view of the low price of butane, but even under these conditions the latter is competing with cheap electric current—for instance in Oregon. To prevent liquefaction in the gas lines at low temperatures, certain precautions can be taken, including dilution with air and addition of propane, which appear to afford protection down to temperatures as low as —20 deg. F. Dilution is accomplished automatically by specially designed apparatus.

As a chemical reagent, butane is already being used for case-hardening. Suggestion has been made that it be used as a reducing agent for producing sponge iron, it possibly being superior to methane, which has been tried for this purpose. Here one should note that although gases like methane have a great potential reducing power, the reduction in the initial stage is too slow to be of commercial value. With respect to methane, the work of the U. S. Bureau of Mines has demonstrated that the reduction assumes a satisfactory rate only after the first stage has been accomplished and considerable quantities of carbon monoxide and hydrogen

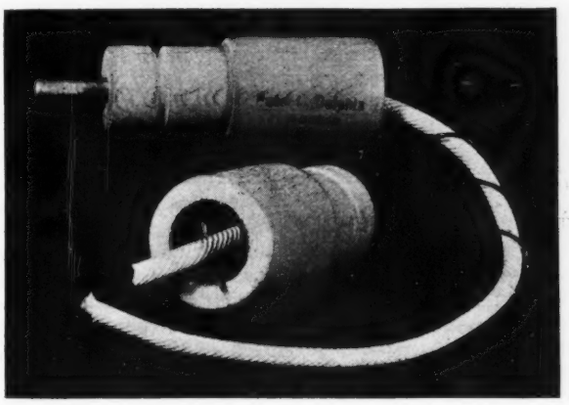
have been formed. With butane the result may be similar, but the general impression is that the initial velocity of the reduction is greater with it than with methane. Reduction of the copper in copper ores in a similar way has often been proposed. It should be easier than the reduction of iron, and could be used, for instance, preparatory to ammonia leaching when economic circumstances permit. Attention has recently been given to the reduction of zinc by methane in special retorts. Substitution of butane for the latter would seem a possibility, should the process succeed. The suggestion has also been made that if, in place of poling, a way could be found to introduce butane into a copper refining furnace so as to provide the necessary agitation, the dissolved oxides of copper could be reduced to the metallic state more cheaply.

A metallurgist has proposed that gases other than air and carbon dioxide be used to create the buoyancy of the concentrate in flotation work. Butane might conceivably be used for this purpose. If the gases so used could be recovered, flotation by gaseous reagents might become important.

Butane has also been used as the basis of various refrigeration schemes, the procedure being to allow the liquid to evaporate, whereupon it absorbs much heat and produces a marked cooling effect.

■ ■ ■

Misfire Hazard in Blasting Reduced by Simple Safety Device



Two views of an interesting blast-protection device, which, it is claimed, greatly reduces the risk attendant upon blasting

Representatives of metal- and coal-mining operators, insurance companies, and explosive manufacturers, as well as of the U. S. Bureau of Mines, recently witnessed a demonstration of an interesting blasting accessory in the Ingersoll-Rand experimental mine which adjoins the rock-drill manufacturing plant of the company at Phillipsburg, N. J. The object demonstrated was the so-called Delphia blast-protection device, designed to lessen, if not wholly to abolish, the hazard from misfired shots in blasting with dynamite and black powder, and to increase the efficiency of blasting. Although new to mining practice in the Americas, it is said to be extensively used in Germany and in other Continental countries. As much as 50,000 bd.ft. of lumber is said to be cut up per day for making it. The Prussian State Government and the Hungarian Government are reported to have prescribed its use in certain blasting operations. The inventor is Edmund Karolus and the manufacturer the Delphia A.G., of Berlin. The device is patented in all civilized countries, the United States patents being No. 1,588,427 (June 15, 1926) and No. 1,683,900 (Sept. 11, 1928). Howard E. Perry, of Portland, Me., president of the Chisos (quicksilver) Mining Company, of Terlingua, Tex., who is interested in the device, arranged the demonstration.

The primary purpose of the Delphia device is to prevent misfires, and if despite precautions a misfire does occur, to make possible the quick, safe withdrawal of stemming and primer and the reloading of the hole.

Inasmuch as misfires are most likely to occur through tearing cap, fuse, and primer cartridge apart in loading, the new blasting device is designed to hold the detonating cap and fuse firmly connected with each other and with the stick of powder into which the cap is inserted, during loading and tamping. It similarly holds an electric exploder in proper place. The inventor has also devised a tool for removing the stemming and the Delphia device with the attached cap and fuse (or wires) from a misfired hole without accidentally exploding the charge.

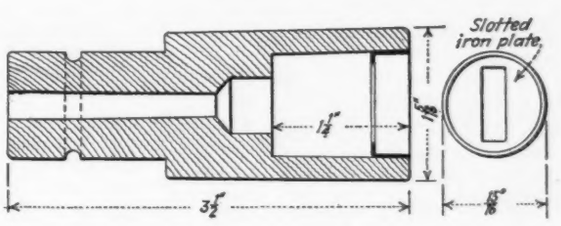
As accompanying illustrations show, the device described is a shaped roll or spool of wood, having a hole through its center lengthwise, which at one end is just big enough in diameter to hold a detonator cap. The fuse or wires attached to the cap pass through the roll and out the opposite end where the hole is wider. Where cap and fuse are used, the fuse is first threaded through the roll from its wide end, and the cap is slipped over the end of the fuse without crimping, after which the cap and fuse are carefully adjusted with the fingers so that the cap projects part way from the small bore. The diameter of this hole should be just large enough to permit the cap to be fitted into it snugly without danger of slipping out. To hold cap and fuse in place a soft iron wire attached to the larger end of the roll is wound around the fuse, as shown in one of the cuts.

When loading a hole, the cap, fuse, and wooden safety device, thus prepared, are inserted in the end of the last stick of powder, inside the wrapper, so that the cap is embedded in the powder. To permit this, the end of the wrapper is opened. The wrapper is then tied fast around the end of the safety device with a piece of string. When this stick of powder has been tamped in place the stemming cartridges are inserted behind it and tamped, after which the hole is ready for blasting.

Where electric exploders are used, to wind an iron wire around the exploder wires would obviously be inadvisable. So in this case the small bore of the roll is tapered slightly toward the inside and the wire is omitted. This taper permits the cap to be pushed only part way into the bore, where it sticks. The sectional drawing shows such a roll.

The special tool devised to remove stemming, roll and cap from a misfired hole consists of a steel rod having an auger at one end and a handle at the other. With it the stemming is first bored, or scraped, out. When the tool reaches the roll, a smaller, threaded screw point on its tip enters a slot in an iron plate sunk in the end of the roll (see illustration) and, engaging with the plate, permits the roll to be drawn out.

Additional advantages of using the



Longitudinal section of blast-protection device. This model is intended for use with electric exploders. Its distinguishing features are described in the text.

blast-protection device, other than those of reducing the number of misfires and of making possible the recovery of the cap and the reloading of a misfired hole, which have been mentioned in preceding paragraphs, are:

1. Measures usually prescribed for dealing with misfires, such as drilling a new hole parallel to the first and of the same depth and blasting it, become unnecessary. These measures involve a loss of time as well as certain risks of their own, such as the danger of drilling into the charge, and in any case they waste money by duplicating work already done.

2. A hole can be tamped firmly without fear of accidental explosion, inasmuch as, when it is being loaded, the wooden roll will always be between the cap and the collar of the hole, thus protecting the cap. The roll holds the cap centrally in the drill hole parallel with it so that it cannot be turned at an angle and forced into the rock wall of the hole when tamping and perhaps prematurely exploded. Premature explosion of the cap under such conditions, if the hole misfires and the cap has become exposed, may produce an open flame which, in a coal mine, may cause a gas or dust explosion.

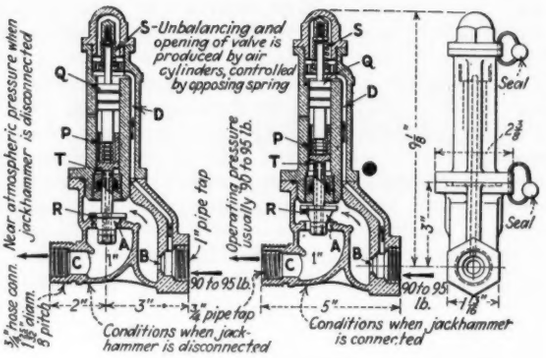
3. Blasting is made more efficient by improving the tamping. Tests made in Prussia are said to indicate an increase of 7 per cent in blast efficiency thus obtained.

4. Crimping of cap on fuse is unnecessary. This obviates the personal risk attendant upon using pliers, hammer or teeth; also the risk of cutting or interrupting the powder train in the fuse, or of admitting moisture to the fuse by breaking the covering.

When loading wet holes, the cap-fuse connection can be waterproofed readily by filling the wide end of the wooden roll with clay.

The device can be used for all kinds of blast shots irrespective of the explosive used.

Since the use of the Delphia device was made obligatory in Prussia, loss of life from misfires is reported to have ceased. In the United States, about 300 deaths a year are reported to occur in handling misfires.



Sectional view of automatic air-control valve See text for references to letters.

Valve Prevents Air Waste

To prevent waste of air in operating jackhammer drills and other pneumatic tools an automatic air-control valve, which permits definitely limited quantities of air to pass at various pressures, the quantity passing varying directly as the pressure, has been developed by the research department of the Jeddo-Highland Coal Company. It will be made by the Yarnall-Waring Company, Philadelphia. With modifications it can also be used in sectionalizing water lines.

The valve, placed near the end of the air line, consists of a globe valve body of unique design and a separate operating mechanism. These fasten together and are sealed to prevent tampering.

In operation, an orifice in the valve body is set for a given size of jackhammer or other tool, and the springs adjusted to the flow. The valve is then sealed. When the miner connects his hose and has a tight line beyond the valve, the valve automatically opens, permitting just enough air to pass to operate the tool. Pressures on top of piston Q (see accompanying sketch) and under piston P are equal, allowing disk A to open by upward lift of spring S. When the jackhammer is disconnected, pressure beyond orifice B drops. Pressure at T drops also, but full line pressure is exerted on top of piston Q through passage D, thus closing disk A. When the jackhammer is again connected, the passage of air through hole R in disk A restores full pressure in space C and at T. As soon as the tool or drill is disconnected or when a pipe or hose breaks, the valve closes automatically, preventing air waste.

The Jeddo-Highland company has installed seventeen of these valves in its Ebervale mines. Here 560 cu.ft. of air per minute is available to operate seventeen jackhammers and one tugger hoist, a maximum demand of 1,400 cu.ft. of air per minute. Before the valves were installed, the chart indicated consistently low, variable pressure in the mains. Since they were installed, the pressure has been consistently higher and less variable. In addition, it is said, the safety valves on the compressor have been lifting, something previously unheard of. This uniformly high pressure, permitting the miner to drill efficiently at all times, has eliminated any possible objections he might otherwise have had to the valve.

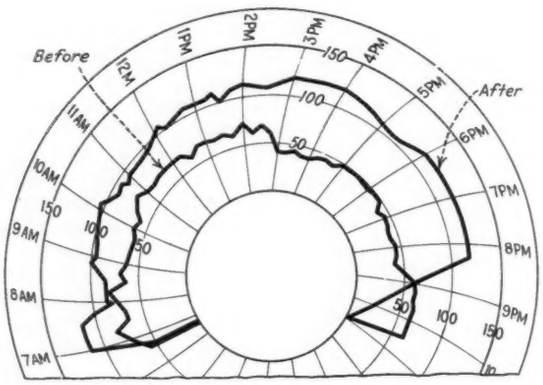
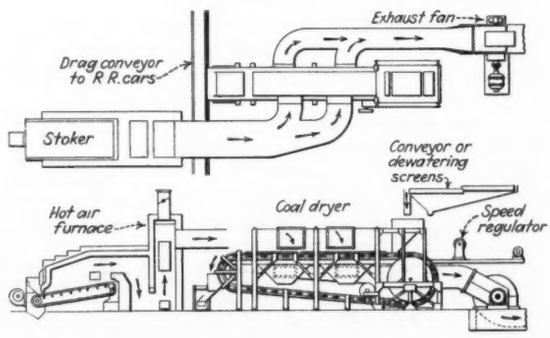


Chart showing air pressure on compressor before and after automatic air-control valve was installed.

Something New in Dryers

Operators engaged in drying or faced with drying problems will be interested in a new continuous dryer, offered by Oliver United Filters, Inc., and said to use established principles of drying in a new way. This machine is known as the Dwight-Lloyd-Oliver dryer, or for



This new continuous dryer has a straight-line traveling bed on which the material is dried by passing hot air or gases through it.

the sake of brevity, the "D.L.O." dryer. It is built by the Sintering Machinery Corporation. Wet material is dried on a straight-line traveling bed, by means of the controlled passage through it of heated air or gases. The dryer is said to reduce moisture content at an operating cost hitherto considered unattainable. Dusting and breakage of materials do not occur. Low fuel rates are obtained by this economical way of applying heat to the charge. Power consumption is less than with rotary dryers of similar capacity.

Feeding Crushers

Two new types of constant-weight feeders for crushers (Type E and Type F) have been introduced by the Hardinge Company, York, Pa. These machines have been developed to take feed as large as 2 ft. in diameter and to prevent clogging in the bin. They may be had with either rubber belt or armored apron. Like all other feeders that belong in the same class, these machines are measured by weight not volume.

■

The Hot Miller—Novel Equipment for Drill Sharpening

An interesting piece of equipment, known as the Morgon hot miller, has been developed for cutting, sharpening, and milling rock-drill bits. The sponsors, Neilsen & Svenningsen, 347 Madison Ave., New York, claim that any shape and size of drill bit can be made with it. The process of making a finished drill bit is as follows: 1. Steel is heated in the furnace. 2. Upsetting of preliminary shape in pneumatic sharpener. 3. Slight reheating. 4. Milling on the hot miller. 5. Punching out the center hole by hand if hollow steel is used. The machine, it is said, does not replace existing drill-sharpening machinery. Instead, a definite saving in wear and tear of the existing sharpener is said to be effected because, when the hot miller is used, the sharpener has only to upset enough metal for the miller to make into a finished bit.

Time required for making bits is indicated in the following figures, said to have been obtained in operation:

In making a 4-point cross bit or a 6-point rose bit on hollow hexagon steel, 7 sec. were taken for upsetting the preliminary shape on the pneumatic sharpener (Leyner No. 50); 25 sec. for slight reheating; 45 sec. for milling on the hot miller, divided thus: 15 sec., calibration of periphery; 15 sec., forming of the cutting edges; 10 sec., wing forming (grooves), and 5 sec., finishing off (totaling 45 sec.); and 10 sec. for punching out the center hole by hand.

The slight reheating mentioned is a precautionary process to insure that temperature does not fall below specified limit during milling and thus cause undue wear of the milling wheels.

Inasmuch as upsetting on the sharpener and the slight reheating proceed at the same time as the milling and hole-punching, the rate of production of new bits from bar stock is one bit per 55 sec. Rate of production working in conjunction with an ordinary sharpener is said to be as great as, or greater, than that of two sharpeners working together. As many as 600 new bits are said to have been made on one hot miller in a nine-hour shift.

Labor distribution for a hot miller and a drill sharpener working together would be: One man for the miller and one for sharpener and furnace. The sharpener being in use for 7 sec. only during the making of each bit, its operator can take care of the furnace also. If two ordinary sharpeners are in use, three men are required, one for each sharpener and one for the furnace.

The miller is said to make a power or air saving possible. It requires an average maximum of 2 hp., or 16 hp. hr. per 8-hr. shift. The sharpener in this instance, working at only part of its maximum capacity, takes but a fraction of the power required at maximum operation when making finished bits. The power saving for the miller-sharpener combination is said to be approximately 0.33 hp.-hr. per bit made.

As for maintenance, 100,000 to 120,000 bits can be made, it is claimed, before the milling wheels require resharpening, this provided the bits are milled between 1,500 and 1,700 deg. F. and the wheels are kept running at proper peripheral speed. The life of the wheels is given as 600,000 bits under best conditions.

The hot miller was developed in France and is being made in England. It has been introduced on the Rand, in South Africa, and in coal fields in Great Britain.

For Sharpening Well-Drill Bits

A machine for sharpening well-drill bits which, it is said, will permit a saving of 50 to 61 per cent in drilling costs and 50 per cent in sharpening costs, has been announced by Ingersoll-Rand Company, New York. It is known as Model WD. With it new bits can be forged on blanks in two or three heats, and dull bits redressed in one heat. The bits are said to be perfectly formed, concentric and accurately gaged. The sharpener embodies the same fundamental principles of design and operation as those characterizing the other sharpeners of the company.

Eimco-Finlay Loader's Performance Better Than Stated

In the November issue of *Engineering and Mining Journal*, pages 592-3, in an article on the performance of the Eimco-Finlay loader, made by Eastern Iron & Metal Company, Salt Lake City, Utah, the following printer's error was made with respect to the installation in the United States company's mine: In the third column on page 592 the statement "Loading cost for this period was 22.525c. per ton, as compared with 13.475c. for hand shoveling" should read: "Loading cost for this period was 22.525c. per ton, a saving of 13.475c. per ton, as compared with hand shoveling."

With respect to the North Lily installation, the ore handled runs 20 cu.ft. to the ton, broken, and not 10 ft. as stated. The time required to make a loader operator expert is two to four weeks and not "two to four months."

Additional interesting facts regarding the North Lily installation have been received, as follows: Air consumption tests were made with a Toolometer. Counting time out while loader motors were idle, the air consumption was 45 cu.ft. per minute, when loading cars at the rate of 30 per hour. A portable one-car switch, which slides along the tracks and is kept conveniently close to the face, makes possible quick car changes and speeds up loading at least 25 per cent.

■

Industrial Notes

Mine & Smelter Supply Company has announced the appointment of Harry W. Newton as district manager of its New York office. Mr. Newton's experience has included sales development and engineering work with The Dorr Company and the Nordberg Manufacturing Company, and management of the Conservation Bureau of the Portland Cement Association, this preceded by extensive operating experience in mining properties in the United States and Canada.

■

BULLETINS

Wire Rope Slings. John A. Roebling's Sons Co., Trenton, N. J. An 85-page illustrated pamphlet.

Excavators. Bucyrus-Erie Co., South Milwaukee, Wis. Folder. The 16-B ½-yd. excavator, convertible to shovel, dragline, crane, clamshell. Pp., 8.

Underground Loader. Eastern Iron & Metal Co., Salt Lake City, Utah. Bulletin. Eimco-Finlay Loader. Pp. 12.

TRENDS AND DEVELOPMENTS IN THE INDUSTRY

IN MOST QUARTERS of the mining industry the work of plant rehabilitation and modernization is a feature of normal operating procedure. Specific facts in respect to current trends, obtained by making inquiry among mining companies, are presented herewith. They indicate that in the mining industry as a whole a large amount of such work is being done and that much money in the aggregate is being spent. Names of the companies giving this information have been suppressed because of the reticence of some of the operators.

Company No. 1, mining lead and zinc, has spent about $100,000 thus far in 1932 in improving its mines and smelter, and contemplates spending $250,000 in its pigment plant in 1933, if business indications warrant. Its work this year has consisted in revamping one coarse-crushing plant by installing a new gyratory and a new cone crusher, and in increasing its warehouse storage capacity at another point. Company No. 2, mining sulphur, has made minor changes in its plant amounting to $75,000. This plant, it may be added, is new, having been completed in 1930. The management, however, considers it necessary to maintain it in the best possible condition. Company No. 3, mining lead and zinc, plans installing additional pumping equipment. Company No. 4, mining gold, has, in the last twelve months, spent $395,300 on equipment used in plant rehabilitation, and is now spending $250,000 on new construction, of which $114,000 is for equipment. The $395,300 spent on rehabilitation does not include ordinary operating replacements or an important quantity of equipment made in the company's foundry and shops. The $395,300 was expended thus: Two electric shovels, $39,000; conveyors, $6,000; gasoline shovel, $13,000; waterline pipe, $18,000; valves, $1,000; power-plant equipment, $25,000; shop equipment, $1,200; pumping equipment, $5,000; treatment plant roofing and equipment, $15,500; hoisting rope, $11,000; electric hoist, replacing a steam hoist, $96,000; steam-electric generating equipment, $114,000; and miscellaneous equipment, $5,600. The $114,000 being spent on equipment called for by new (plant) construction will be distributed thus: Structural steel, $23,000; other building material, $13,000; machinery and equipment, $48,000; redwood vats and tanks, $19,000; tile, $2,000; and pipe, $9,000. This company's new plant construction during the next two or three years is expected to be well above normal.

■

Recently, at a conference of the Banking and Industrial Committees of the twelve Federal Reserve districts of the United States, a Committee on Industrial Rehabilitation was organized in Washington, under the chairmanship of A. W. Robertson, with authority to devise and put into action a program to accelerate the modernization and rehabilitation of industrial plants. The primary purpose was to bring about a speedier recovery of business in general by stimulating the recovery of the capital-goods industries, or industries producing plant machinery, equipment, and facilities. The almost complete collapse of these industries, caused by the failure of plants that normally consume their products to make normal replacements of equipment, or to make applications of improved equipment, is held responsible for much of the present unemployment. To aid plant executives in determining where economies can be made through modernization, the committee has prepared a check sheet which suggests a convenient method of making a systematic plant survey. Copies can be obtained by addressing the Committee at 435 Seventh Ave., Pittsburgh, Pa.

■

Company No. 5, mining gold, is making changes in the flowsheet of its 800-ton mill, involving an estimated cost as follows: Changes to 800-ton mill, now, $65,000; mill enlargement in 1932, $270,000; structural rehabilitations, $100,000. Company No. 6, mining gypsum, has spent $150,000 on equipment and material for rehabilitation in the last twelve months. In the next six months it plans to spend $150,000 on rehabilitation, and $250,000 on modernization and new construction, both sums being exclusive of labor costs. Company No. 7, mining gold, has replaced a tramway cable at a cost of $10,000. Company No. 8, mining gold, has spent about $100,000 excluding labor, in modernizing a mine surface plant. This includes a new compressor, alteration of hoist, headframe, and pumping equipment. At another plant, it has done a similar job, costing $25,000, excluding labor.

Company No. 9, mining gold, has reopened its mine and built a 50-ton pilot mill, spending $30,000 to $40,000 thus far. The mill is equipped with two jaw crushers, a 5x4-ft. ball mill, a duplex classifier, a thickener, and a 6-cell flotation machine. Next summer, if results warrant, a larger mill will be installed. Company No. 10, mining gold, has in 1932 built a 50-ton plant including ball mill and flotation plant. Company No. 11, having a relatively new plant treating a copper-zinc-silver ore, states that it has been making numerous small changes and innovations, to lower costs and improve operating efficiencies. The total is considerable.

Company No. 12, mining gold but now shut down, expects to reorganize soon and resume, employing up to 100 men. Approximate expenditure for rehabilitation is estimated at $10,000. Company No. 13, treating an ore containing principally gold and silver, has remodeled its mill, installing flotation. Company No. 14, mining an ore containing lead, zinc, iron, and gold, has spent approximately $250,000 acquiring an old mill, rebuilding and remodeling it to 100 tons' capacity. The flowsheet comprises ball mill, a tube mill, and tables, with amalgamation of the lead concentrate, cyanidation of the zinc-iron concentrate, and cyanidation of the table tailing. New construction includes a 12,600-ft. wire-rope tramway, an 800-hp. automatic hydroelectric power plant, 16,000 ft. of flume, and 7 miles of power lines. The gold refinery includes an oil-fired tilting crucible furnace.

Company No. 15, mining gold and having a mill that is only five years old, has installed $9,000 worth of new equipment in 1932, in addition to replacing worn-out rock drills.

PHILIPPINE ISLANDS

Gold Industry Advances and Protects Itself. Formation of the Philippine Gold Mining Association recently was for the purpose of protecting investors and benefiting from the cooperation to be secured by producers. Discoveries of gold and iron ore have been made in the Province of Nueva Vizcaya, but progress is mainly with established companies, of which the most important is Benguet Consolidated, the subsidiary of which, Balatoc Mining, is increasing plant equipment to meet additional demands. Itogon Mining is also adding to its facilities for the production of gold. Baguio Mining has recently added new mining equipment, and is planning a metallurgical plant. Benguet Exploration has finished experimental work and will commence production operations with a small leaching plant. Salacot Exploration has carried underground development to a point that warrants erection of reduction machinery. Eighteen companies are operating in the Baguio district. Increase in gold production is imminent.

SOUTH DAKOTA

Additions to Cyanide Plant. Homestake Mining, operating at Lead, S. D., is adding about 35 per cent to the capacity of its No. 1 sand plant, to provide extended treatment for the deep-level ore now going to the mill. A separate building for the new redwood vats is being erected, dictated by topographical conditions, and classifying and fine-grinding equipment is also being added.

CALIFORNIA

Activity Centers on Gold Output. Northern California Mining is opening the old Morrison and Carlock mine in Quartz Valley. A force of men is employed pumping out and cleaning the lower levels. . . . Feldman and Nieto placers, at Atolia, have been consolidated under leasehold, and will be operated by Atolia Rand Placers. According to a report by H. H. Morse, consulting engineer of Monrovia, about 1,800,000 cu.yd. of gravel, with a total gold content of about $1,300,000, is available. The placers are to be worked by a 1,000-cu.yd. dredge, equipped with centrifugal separators. . . . Sailor Boy mine, on Soledad Hill, near Mojave, is being equipped with surface and underground machinery. . . . Gold Bar Dredging has placed its dredge in operation on Trinity River, near Lewiston. The company controls 1,000 acres of placers extending along the river for five miles. Gravel yield averages 18c. per cubic yard. . . . John H. Reeniger and associates, of Los Angeles, have acquired the Oro Fino gold mine, near Auburn, and intend to exploit the reserves of ore said to exist. Erection of hoisting machinery and installation of pumps has commenced. Oro Fino was for many years the largest producer in Placer County. The Taylor and Rosencranz mines, also in Placer County, acquired by Los Angeles interests, are to be rehabilitated. E. Geisendorff, engineer, is in charge of dewatering the old workings. Diamond drilling has revealed ore in the Rosencranz property. . . . Eldorado-Alpine Mines announces completion of a 200-ton mill at the Alpine mine, near Georgetown. About 100,000 tons of milling ore is available for extraction. . . . Installation of a 200-ton mill, involving an expenditure of about $150,000, on the Shasta View gold group, near Redding, is scheduled to be completed before the end of December. The Potosi Mine, ten miles west of Redding, patented by the late Senator Jones in 1875, is again in operation. A 75-ton mill has been built recently. . . . Mining and development work has been resumed by Idaho-Maryland, operating at Grass Valley, with 200 men employed. Work recently completed includes the rebuilding of the hoisting plant and the retimbering of a portion of the shaft. Both the Idaho-Maryland and Brunswick mills are to operate as soon as contemplated changes and repairs have been made. A central milling plant is to be built within a year. . . . California Mines, operating the Bald Rock mine near Oroville, has completed retimbering the shaft and erection of a new mill, which is to be placed in operation at an early date. . . . A stamp mill is being installed on the Rawhide mine by J. Moyle Lloyd, A. E. Porr, and W. Rogers, who are working small veins east of the main lode north of the property, and another by Ruby Mining at its property on the Breckenridge Mountains, 30 miles east of Bakersfield.

Quicksilver Mines Reopened. The Alpine mine, in the Hernandez district, San Benito County, has resumed operations, to develop a strike of cinnabar recently discovered while cleaning out old workings. The new vein is of a nature similar to that used as the main source of ore supply when the mine was in operation during the War. It assays about 4 per cent mercury. The property is equipped with a plant, but only sufficient ore is being treated to keep three retorts in operation, to pay labor, and expenses of development. Additional ore is being blocked out, and when prices improve operations will be resumed on a large scale.

A 50-ton flotation plant is being installed by Consolidated Metal Mines at its Yucca Grove property, San Bernardino County. Development to a depth of 290 ft. has disclosed a considerable tonnage of gold ore. . . . Electrically driven equipment has been installed at the Consolidated Gold & Metals property in Plumas County, owned by the J. R. Walker interests, of Salt Lake, and a power line has been extended from the Walker mine of International Smelting. Mine development is in progress.

UTAH

A. S. & R. Smelter Resumes. A blast furnace was blown in at the Murray smelter of American Smelting & Refining on Nov. 15, and 250 men went to work. Sufficient ore has been stored in stockpiles to assure continuance of operations until April, 1933, according to W. J. O'Connor, manager. . . . A shoot of high-grade silver ore has been opened up 1,200 ft. from the shaft of Park City Consolidated at Park City, on the 800 level, according to J. J. Beeson, general manager. The ore, which has been followed up for about 85 ft., is from 5 to 8 ft. wide and assays from 30 to 60 oz. silver. Reconstruction of the Utah Delaware tramway terminal at Bingham, destroyed recently by fire, will be completed soon. During rebuilding of the snowsheds and the surface plant, ore is being put down a hopper to the Carr Fork road and trucked to the nearest railroad terminal for shipment to the Tooele plant of International Smelting. . . . Two drifts are being advanced on the 600 level of the Park Standard mine, at Park City, adjoining Park Utah. That to the northwest is following vein matter carrying from $1.65 to $2.20 in gold and that to the west 14 ft. of material assaying $3.35 in gold. Both drifts are in the Ontario quartzite.

LOUISIANA

Frasch Plant in Operation. Production has begun at the Lake Peigneur sulphur deposits near Abbeville, La., under control of Jefferson Lake Oil. The sulphur lies at a depth of from 600 to 850 ft., and the property is served by a new intracoastal canal. Rail connection is under construction. . . . Freeport Sulphur plans extensive improvements, involving construction of power plant and treatment equipment, canals, pipe lines, and mining work necessary for the development to a production stage of a sulphur deposit in Plaquemines parish, about 50 miles south of New Orleans.

THE NORTHWEST

Predominant Interest in Gold. Consolidated Mining & Milling, of Helena, is opening gold veins in the Boulder-Elkhorn district, Jefferson County, according to Hershel Smith, consulting engineer. Work will be under the supervision of Albert S. Hanson, president and general manager, of Butte. Development of the property is interesting, as the east-west vein system, with predominating values in gold, is new in a district where the predominating values formerly were lead-zinc-silver. The gold in the veins of the Basin district to the west mine are found in an east-west vein system. . . . Incorporation of Morning Glory mines has been completed under the laws of Montana, and shaft sinking has started. The property is five miles from Basin, Mont. A compressor, engine, hoist and other machinery have been installed, machine drills have been provided, and boarding house and bunk cabins erected. . . . Virginia City Gold has purchased amalgamation and flotation equipment for erection at the Prospect Group mine, in Virginia City, Mont. The plant will have a capacity of 60 tons daily. . . . An important mining deal in northern Stevens County was closed recently when R. O. Reinertsen and associates, of Everett, Wash., purchased the United Treasury mines, 11 miles northeast of Boundary, adjoining the Frisco-Standard. . . . The Boise-Rochester gold mine of St. Joe Lead, at Atlanta, southern Idaho, is recovering 95 per cent of the gold from $12 to $17 ore. Frank Skeels, formerly of Wallace and for several years in South America, is in charge. . . . Grade of the orebody on the 1,100-ft. level of the Gold Hill and Iowa is so satisfactory that A. H. Burroughs, Jr., president of Talache Mining, owner, is preparing plans for a new mill. . . . Operations of the Hecla Mining yielded a profit in the quarter ended Sept. 30, as compared with a loss in the second quarter. The gross income in the third quarter was $205,624, compared with $164,652 in the second. Total expenses were $190,463 in the third and $187,707 in the second. Respective production figures were: lead—8,913,662 and 8,208,550 lb.; silver—225,-143 and 205,750 oz. . . . Development and equipment of a placer having $75,000 net profit in 45 acres of tested ground, and expectations of 200 acres more that have been tested, have been announced by Key Placer Corporation, operating near Golden, in the Ten Mile district of central Idaho. The estimate is based on reports of S. H. Richardson. Key Placer has laid 4,000 ft. of steel pipe that will carry water for the operation of two hydraulic giants. Sluicing will be resumed March 1. Acquisition of the property began when a car lodged in snow two years ago, and when the four occupants, seeking shelter, followed a light to the cabin of J. P. Kee. Although alone, ill, and 82 years old, Mr. Kee welcomed them and served coffee, but was too sick to do more. On hearing of their search for placer ground and their readiness to invest, Kee described his property. He was ready to surrender to younger hands. A deal was made, and the veteran retired to Grangeville, Idaho, where he now lives.

. . . The annual convention of Northwest Mining Association will be held at Spokane Dec. 15. Plans for the meeting have been changed since it was first announced. It will be entirely under the auspices of the Association, participating organizations having abandoned winter meetings this year. Bluesky legislation, the remonetization of silver, and the furtherance of geological surveys and topographic mapping will be discussed. New legislators and Congressmen will be invited to attend, to learn the position of the mining industry on such matters.

NEVADA

Gold Recovery Plants Nearing Completion. Arizona-Comstock is completing erection of a 150-ton mill, which is expected to commence operations soon on ore from the Hale & Norcross and other mines in the Virginia City district. . . . Nevada Porphyry Gold, operating underground and placer properties, reports excellent results, although the hydraulic work must soon be suspended because of exhaustion of water supplies. . . . The high-grade vein in the Jumbo mine of Goldfield Consolidated, responsible for a yield of about $75,000 during the summer, has passed into the area reserved for and operated by Goldfield Consolidated. The Fitzgerald-Svorcan lease is therefore likely to terminate. . . . Stone Cabin Consolidated is now operating a flotation mill in the neighborhood of Virginia City, and satisfactory mine developments are reported. . . . At National, National Consolidated is embarking on a campaign of diamond drilling. . . . Steam-shovel practice is to be adopted at the property of Comstock Gold, operating at Brunswick, near Carson City, and additional milling plant is being erected. . . . Work has been resumed on the property of Nevada-Montana Mining, at Placeritos, and facilities for an increase in the treatment of placer ground are under way. The property ownership has passed to J. W. McCormick. . . . Sinking of the Rio Tinto shaft by International Smelting at Mountain City, Nev., has reached a depth of 80 ft. below the old workings, or 320 ft. below the surface. The bottom of the shaft is reported to be in ore, although of lower grade than that followed to the 227 level, which averaged from 30 to 50 per cent copper.

COLORADO

Molybdenum Mine to Reopen. The molybdenum mine at Climax, near Leadville, will resume operations at maximum capacity in a few days, according to the "Salida Mail." The mine shut down in August because of the reduction in demand for molybdenum, and the company was reported to have a two-year supply on hand. But orders were received recently, according to report, for eleven carloads of concentrate to be shipped to Krupps in Germany, nine carloads to be shipped to England, and seven carloads to France. A carload was shipped to Japan less than a month ago. This has depleted the two-year stock, and the mine is to be again put in operation. The company employs about 300 men.

COPPER COUNTRY

Future of Mining Industry. The future of the Michigan copper district is apparently secure. Reserves are large, promising new deposits have been located, and 90 per cent of the copper-bearing area is yet to be explored. The chief uncertainty is the price the metal will bring when industrial conditions improve. The Michigan district has experienced periods of low-priced copper in other years, but it was not in any of those times faced with the menace of cheaply produced foreign metal. This menace has been temporarily removed by the 4 cent tariff, but the domestic industry must look to a continuation of protection beyond July 1, 1934, when the revenue bill expires by limitation, if it is to be prosperous after the accumulated stocks of copper on hand are absorbed. . . . Considerable optimism is in evidence in the district as to the finding of additional reserves of copper. Although the remaining copper-bearing territory is covered with overburden of varying depths, which increases the difficulties of finding new deposits, the application of geophysical methods has been found practicable in locating favorable areas. The elimination of lode zones in which there is less likelihood of finding copper will result in avoiding much diamond drilling and drifting. Copper is concentrated in shoots within the veins. Changes in ore grade are due to changes in rock texture, not to depth. . . . The mining industry constitutes half of the total business of the upper peninsular of Michigan. It furnishes the chief market for agricultural and timber industries. This is the principal reason for the State-supported copper and iron research program being carried out by the Michigan College of Mining and Technology at Houghton, in cooperation with the copper and iron industries. Copper costs have been considerably reduced as a result of work done, and progress is being made in tests on coarse-banded iron rocks taken from deposits regarded by geologists as having the most promise of ultimate commercial use. Research will probably find methods of making this low-grade material commercial. If this is done, Michigan can mine iron for hundreds of years. Research is regarded as a State problem, because the State and the public are the chief beneficiaries of the copper and iron mining industries. The greater part of the dollar received by a mining company is paid out for labor, supplies, services furnished, and taxes. Mining companies can transfer their operations elsewhere when ores are exhausted. The homes, farms, and businesses of citizens, however, cannot be transferred. The public, therefore, is considered as having a vital interest in perpetuating the mining industry.

Copper Companies Meet Community Obligations. Calumet & Hecla and Copper Range, the active mining companies in the Michigan copper district, are mining high-grade ore, enabling them to continue operations, on a restricted scale. Work has been provided as a relief measure. Otherwise, the mines would have been closed. The conviction that the companies owe much to their communities is the chief factor in continuing operations as long as possible, even though an average of only three days' work per week per man is provided. . . . Calumet & Hecla is mining only conglomerate rock in upper levels, taking out the rich pillars and backs of old stopes. A considerable tonnage remains. When this source is exhausted, however, yield from the conglomerate will be considerably lower, the rock in territory below the 81 level and to the north of Red Jacket shaft being of comparatively low average grade. An extensive lode area exists, however, and the conglomerate should have a long life as a producer. Although a continuity of high-grade ore occurs in the upper levels, in the lower ground and to the north the copper occurs more in bunches. To the north, the copper content is somewhat better than below the haulage level. . . . At Copper Range, a high yield per ton is being obtained in the Champion mine by close selection. Reserves are being well maintained, and more than half the ground opened is of good or fair grade. By changes in mining methods, and a general cost-reducing program, including cheaper power, Champion has been able to operate during the depression. The showing in the Globe tract continues encouraging. This strengthens Copper Range's position as to the future. The company also has other promising mineral tracts that will be given attention when conditions improve. . . . Total enrollment at the Michigan College of Mining & Technology at Houghton for the first term of the current college year is 561, as compared with 592 for the same term a year ago. About 80 per cent of the students enrolled are residents of Michigan, and more than 60 per cent live in the upper Peninsula.

CANADA

Cyanide Regeneration Process. Noranda Mines has developed a large tonnage of ore with payable gold content that cannot be treated economically by concentrating the sulphides, and gold is being lost in the present concentrator. As consumption of cyanide in treating this ore by the usual method is unprofitable, Noranda metallurgists have developed a unique method for regenerating the cyanide. Equipment is now being added to a 500-ton unit of the concentrator to cyanide the ore by this method. If this pilot plant gives good results, the whole mill will be so equipped. . . . Meantime, an additional 500-ton unit has been added to the concentrator, giving it 1,500 tons' capacity. Another unit, now under construction, is expected to bring the mill to 2,000 tons' capacity in January next. This tends to increase the smelter output, as the concentrate is somewhat higher in copper and much higher in gold than the direct smelting ore. The ore reserves continue to increase, and the average grade to improve slightly. The adjoining Chadbourne gold-quartz deposit is showing a large amount of low-grade ore. . . . Ashley gold mine, discovered just over two years ago, shipped its first bullion early in November. The mill is working at about half rated capacity of 150 tons and the mill feed is at about $20. A winze is being sunk to the 750-ft. level, to open

up two new levels below the four now established. . . . Great Bear Lake silver ore has had its first commercial test. A shipment of 10½ tons to the smelter at Trail, B. C., costing $180 a ton for freight, netted the owners $10,495. The ore yielded 36,230 oz. silver, or 3,450 oz. per ton. The radium-reduction plant at Port Hope, Ont., is now in partial operation and is expected to yield radium salts in marketable form by the end of the year. . . . The McWatters gold find, three miles east of the Granada mine, in Rouyn Township, promises to be one of those rare deposits that will pay from the grassroots. Several mining companies have tried to acquire a share in it, but the prospecting syndicate that opened it up. Prospectors Airways, has arranged financing operations. Two mining companies, Ventures, Ltd., and Huronian Mining & Finance, have substantial interests in the syndicate. The gold occurs in quartz veins and stringers within a shear zone, 60 ft. or more wide and opened at intervals for 600 ft. in length. Several sections within this shear are likely to yield ore, but attention at present is focussed on an 8-ft. vein which shows persistently high values, averaging well over $100, so far as it has been stripped. Visible gold and high assays are found so well distributed throughout the various exposures that a large volume of ore is assured. The showing is in conglomerate of Timiskaming age, as at Granada. Diamond drilling has been started, to serve as a guide for the underground work that will soon ensue. . . . Granada is to have the first Hadsel mill to be installed in Canada, as part of the enlarged mill now in course of construction. A statement by the management says that the present cost of $8 per ton is expected to be cut to $6 in the new plant. This reduction will permit of using a considerable amount of ore, already partly developed, that has heretofore been unprofitable. Lateral development has been commenced at the 1,225-ft. level. It is largely in anticipation of ore to be opened between this and the 625-ft. level that the mill is being enlarged from a capacity of 100 tons to 200 tons or more. The single unit of the Hadsel mill now ordered is expected by the management to have a capacity between 200 and 300 tons.

BRITISH COLUMBIA

Cariboo Gold Quartz Expectations. The Annual Western Meeting of the Canadian Institute of Mining and Metallurgy was held jointly with the Mining Association of British Columbia in Vancouver on Nov. 30-Dec. 2, and was given an international character by the participation of the Northwest Section of the A.I.M.E. The meeting was attended by the President of the Canadian Insitute, Dr. J. A. Allen, of Edmonton, and by the secretary-treasurer, E. J. Carlyle. Technical papers presented included a symposium on gold milling in British Columbia, with special reference to practice at the Premier, Pioneer, and Bralorne mines; an account of recent mineral developments in the Great Bear Lake area, by D. F. Kidd, of the Geological Survey; a series of papers on placer mining and placer mining opportunities in British Columbia; a paper on "Gold Mining in Manitoba," by A. G. McLaren, Inspector of Mines for that Province; an address by Dean Milnor Roberts, of Seattle, on the "Training of Men to Engage in Placer Mining"; one by Paul Billingsley on "An Attempt to Rehabilitate the Nickel Plate Mine at Hedley"; and an address by T. A. Rickard on "The Primitive Use of Metals." A feature of the meeting was an address by the Provincial Mineralogist, John D. Galloway, on lode-gold developments in British Columbia. The value of lode-gold production for 1932 is estimated to have been about $3,800,000, or an increase of 26 per cent over that of 1931, attributable to an increased output from the Pioneer and the coming into production of the Lorne mine, in the Bridge River district. Premier, at Stewart, however, still continues to be the largest individual producer of gold in the province. Mr. Galloway states that the most important lode-gold development of the year has been that of Cariboo Gold Quartz, operating near Barkerville, in the Cariboo district. About two years ago the company decided to drive a long, low-level, crosscut tunnel to determine the persistence of the veins exposed on the surface at depth. During the past year this tunnel, now about halfway to the original objective, crosscut eight veins, most of which give promise of minable widths of good milling ore. Thus, Mr. Galloway states, "In effect this is a new mine as distinct from several promising veins lying some 1,500 ft. ahead of the present face of the main tunnel." He adds, "While drifting and raising on the veins has not yet been undertaken to block out important tonnages of ore, all indications point to extensive ore reserves. In 1933 a substantial output of gold should come from this property." The veins exposed in the underground workings are all cross veins that cut the schistose formation on strike and dip and are quartz-filled fissures that give every evidence of permanence. They pinch and swell in short distances along the strike, ranging in width from a few inches up to 10 ft. The principal mineralization is with massive pyrite, but in addition a soft, gray-black bismuth-lead sulphide mineral, widely distributed in all the veins, occurs sparingly, accompanied by free gold. It is an excellent indicator of high gold values. The gold is directly associated with the pyrite mineralization; and, in part, in a fine state of division, it is free from the pyrite, but apparently most of it is directly associated with it. Power at the mine is generated by diesel engines. A 94-hp. engine and a 409-cu.ft. compressor serve the mine, but another 114-hp. engine and 630-ft. compressor are to be installed in the near future. A 224-hp. engine and generator are now being provided to provide electric power for the mill, immediately below the tunnel portal, an all-cyanide plant. It is expected to treat 75 tons a day, and at small additional cost capacity can be doubled when desired. Tests by the Dominion Mines Branch showed the ore to be nearly ideal for cyanide treatment. By grinding through minus 48 mesh, and agitating for 24 hours in cyanide solution, an extraction of 97.1 per cent was made on ore assaying 1.04 oz. per ton.

Britannia Operations Threatened. The understanding reached in London recently by British consumers and producers on the terms of the preference tariff on Empire copper discriminates against producers in British Columbia, and is not in accord with understandings reached at the Ottawa Conference. Prior to the Conference Canadian base-metal producers at a meeting held in Toronto, unanimously agreed to recommend to the Dominion authorities that an agreement be negotiated with the British representatives to provide for a preference in the United Kingdom on copper of Empire origin, wherever smelted or refined. The inclusion of the word "smelting" in this definition was deliberately made to assure that producers in this province would participate in the market privileges that such an agreement would afford; and the recommendation was based essentially on the recognition of conditions in British Columbia, under which it is not economic to smelt locally the output of the Britannia and Allenby mines, which output must be shipped as concentrate to Tacoma for smelting. The position was thoroughly understood and appreciated by the Canadian Government, and the recommendations of the producers were duly accepted. The Prime Minister evidently presumed that the agreement, as drafted in Ottawa, to have been sufficiently inclusive to protect all Canadian interests, for in a speech in the House of Commons on Oct. 12 he specifically refers to the benefits that this agreement would confer on the copper mines of British Columbia as well as those of eastern Quebec, in that they would be assured of access "to the third largest market in the world for unwrought products." The provisions of the agreement or their interpretation, as now announced, however, exclude from preference Empire copper if smelted abroad. This does not immediately affect the Allenby mine, which has been inoperative for some time past; but it does most seriously affect the Britannia, and it may, in fact, occasion the closing of the mine. Although operating on a curtailed schedule, the Britannia by rotating work has been able throughout the depression of this year to give employment to several hundred men, and is the mainstay of a community of over 1,000 people. The suspension of this operation therefore would be little short of a calamity to the Province. . . . A recent shipment to the Trail smelter was one of silver ore consigned by Eldorado Gold Mines from Great Bear Lake, amounting to about 10½ tons. It yielded 36,230 oz. The freight charges on this shipment amounted to $180 per ton.

NEWFOUNDLAND

Gypsum Deposits to Be Developed. From consular sources the information is available that International Gypsum, Ltd., has been formed to develop gypsum deposits at Flat Bay Brook. The owner of the areas, who is also one of the incorporators of the company, claims that analyses show the mineral to be a hydrous calcium sulphate free of deleterious matter. Plans are to export crushed stone in its raw state and to manufacture and export neat and finished plaster. About five miles of railroad are to be constructed; also a crushing and plaster plant. A New York firm is reported to have offered to take 120,000 tons of rock and 320,000 tons of plasters, and total output is expected to reach 1,000,000 tons per annum.

BRAZIL

Ores in Exchange for Steel. According to a report to the U. S. Department of Commerce, in a recent bid for rails and other track material for the Central Railway, the National Purchasing Committee of Brazil stipulated that preference would be given to the bidder who would take part or whole payment in manganese ore or iron ore. Upon petition from Minaes Geraes, the commission has instructed to make such acceptance a condition to consideration of bids. For this purpose it has been provided that prices will depend on market quotations from the mining industry, varying according to metallic content and other factors of analysis. Prices in gold are preferred. The ore can be delivered in the port of Rio de Janiero or any foreign port, with the former preferred. Transactions would be carried out in accordance with agreements between foreigners and the national mining industry.

VENEZUELA

New Gold Field Reported. The Guiana section of the State of Bolivar has produced about half a million dollars' worth of gold in three weeks, from a recent new find. Subsequent to the discovery, and the enrichment of the pioneers, a gold rush ensued, and the government took prompt action to facilitate the granting of concessions and to preserve order. An airplane service has been established by a French company between Cuidad Bolivar and Tumeremo.

MEXICO

Economic Changes Affect Mining. Two presidential decrees are of importance to the Mexican mining industry. One of these amends the federal mining taxations law of April 27 last, to permit substantial reductions in the imposts on mineral lands; the other authorizes the Ministry of Industry, Commerce & Labor to proceed with the creation of national mineral and hydrocarbon reserves. The new taxes, which apply to lands that produce metallic minerals, are as follows, per year: One to five claims, 6 pesos; six to 50 claims, 9 pesos; 51 to 100 claims, 12 pesos; and 101 or more claims, 18 pesos. Lands which produce non-metallic minerals pay 90 centavos per claim per year. National mineral reserves will be constituted of free lands which produce gold, copper, antimony, selenium, mercury, aluminum, magnesium, coal, iron, sulphur, bismuth, chrome, platinum, minerals that yield radium, rare minerals, graphite, asbestos, cadmium, phosphates, nitrates, tungsten, molybdenum, glucinum, vanadium, and precious stones. Similar reserves will be composed of free lands producing gilsonite and alberite in the hydrocarbon group.

Unemployed Seek Work. Organizations of unemployed miners in the Villa Escobedo, Santa Barbara, Parral, and other regions of Chihuahua state have asked the government to allow them to work, as cooperative units, mines in those regions shut down because their owners found operations unprofitable. . . . Establishment of the six-hour shift in Mexico has been urged upon the government by various labor organizations, to provide work for more men, to apply only to newly established industries such as new mines and new mineral plants. . . . Cananea Consolidated Copper has been ordered by the Federal Board of Conciliation & Arbitration to pay its employees for holidays they took on Sept. 16, Mexico's Independence Day, and Christmas Day of last year, and on May 1, national Labor Day, of this year. In opposing payment, the company asserted that the federal labor law, which provides that these are days of rest with pay, is unconstitutional, and that it violates labor contracts and concessions held. The Board held that the federal labor law is in full accord with the national constitution.

Mining Conditions Improve. Cia. Minera de San Rafael y Anexas, operating in the Pachuca zone, state of Hidalgo, has withdrawn its application to the authorities for sanction to dismiss workers on the ground that operations are unprofitable, in view of low silver prices. In withdrawing the application, the company stated that conditions which prompted the desire to reduce personnel have disappeared. . . . Reports to the effect that Moctezuma Copper has resumed operations at Nacozari, Son., are declared untrue. Considerable prospecting activity in the region is said to have inspired the story. . . . A 35-ton flotation plant, driven by a Fairbanks-Morse 110-hp. diesel engine, and a diesel-engined C.P. air compressor, are among improvements installed by Cia. Minera Los Reyes, subsidiary of the Los Reyes Mines, with headquarters in San Francisco, Calif., and property in the Ures region of Sonora. . . . Cia. Minera y Molinera del Baja California has installed three gold-washing machines near Ensenada, close to the American border in the northern district of Lower California. . . . A daily average of about 350 tons of gold-silver ore is being mined and milled on the property at Rosario, Sinaloa, owned by the Cia. Minas del Tajo. This region has been worked almost continuously since about 1585. . . . The low value of Mexican currency against the dollar is enabling Cia. Minera de Las Dos Estrellas to operate, at full capacity, its mining property at Mineral de Dos Estrellas, Michoacan.

AUSTRALASIA

Discovery in the Interior. An extensive virgin gold field has been found at The Granites, Central Australia, in an uninhabited territory 65 miles southeast of Tanami and 360 miles northwest of Alice Springs. The find has attracted tremendous interest, and four companies with an aggregate capital of £625,000 have been formed to carry out development. One of the companies has despatched three diamond drills and six large rotary drills to prove the outcrops and locate water. Airplanes are being widely used for transport to and from the field. Gold mining companies formed during recent weeks include Gold Options, to develop the Shackell mine, in the Northern Territory, and a lease on the Granites Goldfield, Central Australia; Chapmans Gold, to develop several leases on the Granites gold field; Granites Development, to purchase leases at the Granites; Central Gold, to acquire 85 acres at the Granites; Granites West, to acquire leases at the Granites; Royal Standard, to acquire the Royal Standard group of mines at Woods Point, Vic.; South New Chum, to develop a property at Bendigo, Vic.; Ironbark South, to acquire and prospect a lease at Bendigo; Guildford Plateau, to treat 1,250 acres of alluvial wash near Castlemaine, Vic.; Spring Gully, to develop a lease at Fryerstown, Vic.; Yellow Glen Gold, to carry out development on a lease at Yandoit, Vic.; Binalong Gold Mining, to purchase the Binalong mine in New South Wales. . . . Austral Development has acquired an option over a gold lease at Schist Hill, about 30 miles west of the Granites field. The outcrop is said to be prominent for over a mile. Little or no work has yet been done.

Victorian Government Assists Prospecting. Bendigo gold production for the nine months ended Sept. 30 totals 14,275 oz., an increase of 4,000 oz. compared with the corresponding period of 1931. Widespread prospecting continues throughout Victoria. About 5,000 men are being assisted financially by the government. Men receive grants up to 15s. weekly, and are provided with equipment and explosives.

Gold Output Increased. West Australian gold yield for September was 54,427 oz. fine, which is 16,254 oz. greater than that for September, 1931. Yield for the nine months of 1932 was 448,314 oz., an increase of 93,570 oz. over the 1931 period. . . . Wiluna Gold Corporation reports that new reagents are now being used in the flotation plant, and that recovery has been much improved. The power plant is being increased by two diesel units. With additional power the mill will be able to handle larger tonnages. Extensive development and diamond drilling are being carried on. On the Iron Duke lease, Kalgoorlie, owned by Associated Northern, rich ore carrying telluride has been met in a west crosscut on the 300-ft. level. The ore is said to be from an entirely new lode, and vigorous development is in hand.

Wage Discussions Continue. At Broken Hill, N.S.W., the unions and the companies are negotiating regarding wages. Mr. C. J. Emery, president of the Mining Managers' Association, placed before a conference an offer of £4 per week for a twelve-month agreement, or £4/4/- for a three-year contract, emphasizing the fact that unless wages were reduced, part-time work would become essential. He also stated that only the exchange rate had enabled the companies to continue in production. This offer has been rejected by the miners, who want a minimum rate of £4/10/- weekly. Further negotiations are progressing. Broken Hill Proprietary, which has options on a gold mine at Stawell (Vic.), and on the Hannan's North, Enterprise, and King of Creations mines, at Kalgoorlie, is negotiating for the purchase of the North Kalgurli mine, which has ore re-

serves estimated at 275,000, assaying 8 dwt. and upward.

Queensland Producer Resumes. At the Mount Morgan gold-copper mine, in Queensland, the flotation research work being carried on under a grant from the Queensland Government is giving satisfactory results. The plant has been reconditioned and is treating 100 tons of ore daily on one shift; mill capacity is 500 tons daily. Ore reserves include 664,000 tons assaying 6.15 dwt. gold and 0.9 per cent copper, to be mined by open cutting; and at least 8,000,000 tons, assaying 4.4 dwt. gold and 1.72 per cent copper, to be mined underground. H. W. Gepp is acting as consulting engineer in behalf of the Queensland Government.

Progress in New Guinea. Bululo Gold Dredging treated 195,100 cu.yd. for the four weeks ended Sept. 9, and recovered 3,592 oz. of bullion valued at 48,492 gold dollars. Dredging is being confined to the eastern side of the property, and the dredge is now digging low-grade wash outside the bored area. Because of particularly low dredging costs, this ground can be treated profitably. Effort is being made to develop the sulphur deposits in New Guinea, and to export the sulphur to Australia, whose requirements are 175,000 tons of sulphur annually. The deposits cover an area of twenty acres, and contain at least 1,500,000 tons of 97 per cent sulphur.

EUROPE

Lead-Zinc Operation Successful. Trepca Mines, operating at Zvecan, Yugoslavia, will pay a dividend of 7 per cent on Dec. 13, making 10 per cent for the year. Trepca has proved a successful mining enterprise. Its capital was all floated at par. With the money so received the property was purchased, the mine developed and equipped, and the mill erected, the capacity of which has since been increased 50 per cent, the outlay being paid for out of earnings. Its annual output is at the rate of about 46,997 tons of lead and 45,500 tons of zinc, in concentrate form. From the time it began producing, Trepca has been obliged to sell its concentrates at about the lowest prices for lead, zinc, and silver in history. The ability to earn substantial profits despite such low prices is due to favorable natural conditions, rich ore that yields readily to treatment, and exceptionally good management. Trepca's Stantrg mine is developed three or four years ahead at present. The ore is mined through adits and sent to the mill by rope tramway, and the concentrates sent by rail directly from the plant to Salonica, thence going by water to the smelter. In addition to its Stantrg mine, Trepca has a large area containing many ancient workings, remaining to be explored and developed. Attention is now being given some of these prospects.

Boliden Adds to Plant Facilities. According to the United States Commercial Attaché at Stockholm, a special committee of the Riksdag recently made a study of the Boliden mines, in northern Sweden. The report has not been published, but is said to be favorable. The mine produces about 800 metric tons a day for refinement at a near-by smelter. It is expected that additional plant will bring the quantity treated to 1,000 tons a day by the end of 1932. The electrolytic smelter has so far had only trial runs; the quantity of copper available has not allowed economical operation. The blister copper has been shipped to foreign smelters. In 1933 the copper will be refined at the plant, and the handling of 1,000 tons of ore a day is estimated to meet about 50 per cent of the Swedish demand. Average copper content indicates 20 tons of pure copper a day from 1,000 tons. The blister copper from the smelter contains about 99 per cent copper and 1 per cent silver, gold, and impurities. Impurities are removed in the electrolytic process. Average gold content is 20 grams per ton of ore, or about 20 kilos a day, with twice as much silver. When regular operations commence, the question of purchase of this gold by the Swedish Riksbank will come up for decision. Large quantities of arsenic add to the value of the production. Over an area of about 7,000 sq.ft. the ore is said to contain a high percentage of gold, which increases with depth, reaching as much as 80 grams per ton at a depth of 400 ft.

Gold Ore in Ancient Mine. Gold ore has been reported discovered in Skouriotissa, Cyprus, and work is under way to mine it, according to a report from the United States consul at Port Said, Egypt. Some of the ore is reported of high value, but the quantity so far discovered is small. A statement from the general manager of Cyprus Mines Corporation is as follows: "The Cyprus Mines Corporation has known of the existence of gold ore on the marginal areas of its principal pyrites for several years. Some of the assays were high, but the deposits of ore so far as it has been able to discover are narrow, small, irregular, and uncertain, so that it has not been considered worth while to spend very much on development as long as shipments of pyrites were good. During the last few months the depression in the copper and pyrites markets has diverted the attention of the corporation to more careful examination and the development of the gold ore deposits, and this organization is now doing a fair amount of work to determine whether orebodies of commercial imports can be proved. Up to this time the company has not demonstrated the existence of the ore in sufficient quantities to assure commercially profitable operation. However, it is continuing in the hope that larger quantities will be found."

AFRICA

1933 Zinc Output Sold. The electrolytic zinc plant of Rhodesia Broken Hill Development, in Northern Rhodesia, will resume production Jan. 1. Reduction of freight rates and the improved position of the metal make resumption advisable. The plant will not reach full capacity till about April 1. Production for 1933 is estimated at 18,500 tons, and it has already been sold. Tanganyika Concessions, the British company that originally formed Union Minière du Haut-Katanga, some time ago acquired options on large holdings in the new Kakamega gold fields, in Kenya. Sir Albert Kitson has been examining the Kakamega area for the Kenya Government, and has issued a favorable interim report. He believes it will develop into an important field, and recommends further trenching and deeper sinking. The sinking so far done has revealed a tendency for the reefs to widen with depth. Up to the end of September 4,270 oz. of gold had been recovered in the field, including one nugget of 9¼ oz. and another of 8½ oz. Sir Albert is quoted as later saying that Kenya's mineral resources are not confined to Kakamega; that he has seen specimens from several different parts of Kenya, and believes there are many gold districts in the region.

Northern Rhodesian Copper Costs. Roan Antelope's report for the year ended June 30, 1932, confirms the reported low cost of producing copper, and also that the company has contracted to sell a large portion of its output for several years ahead. During the September quarter this year the cost of delivering blister copper to European buyers was £23.013 per long ton, exclusive only of interest and depreciation. D. D. Irwin, general manager, says: "It is interesting to note that underground development and stoping have indicated that the original ore reserve estimates were conservative, both as to tonnage and grade," adding that the practice of making a high-grade concentrate, established when concentrate was being shipped to America for smelting, was being continued. Savings in smelting and fluxing expense more than offset the slightly increased losses of copper in tailing. . . . A large interest in Roan Antelope stock has been acquired by Gold Fields Rhodesian Development, it was stated at the meeting of the latter company held Nov. 10. The chairman gave as the reason that Roan Antelope is making substantial profits despite the low price of copper. He said the investment in Roan stock had been made at lower prices than those now prevailing. . . . Sir Edmund Davis recently said that "negotiations are taking place which we hope will lead to a settlement of a satisfactory quota for Northern Rhodesia." He referred to copper. This probably means that, because the output capacity of their plants having proved far larger than originally estimated, Rhokana and Roan Antelope will present a claim for an increased quota when curtailment negotiations are discussed.

Belgian Metallurgical Profits. Soc. Générale Métallurgique de Hoboken, in which Union Minière owns a 50 per cent interest, made a net profit of over 38 million francs in the year ended June 30, 1932, paying a dividend of 11.37 per cent. Its works at Hoboken, Rellep, and Oolen produce lead, copper, tin, gold, radium, cobalt, arsenic and metallic compounds. According to the Soc. Générale de Belgique, the leading Belgian bank, most of the Belgian metallurgical works have sufficient orders on hand to employ their labor during the winter. Transactions in the metal markets are broader than they were two month ago, and the prices are steady—even moderately higher.

Suspensions in South West Africa. South West Africa, governed as a mandate by the Union of South Africa, depends mainly on diamond mining and base metals for revenue. Operations of many properties have been suspended entirely or greatly curtailed.

Daily and Average Monthly Prices of Metals

November, 1932

United States Market

Nov.	Electrolytic Copper: Domestic Refinery	Electrolytic Copper: Export Refinery	Straits Tin New York	Lead: New York	Lead: St. Louis	Zinc St. Louis
1	5.025@5.150	4.925	23.350	3.000	2.875@2.900	3.000
2	5.025@5.150	4.875	23.400	3.000	2.900	3.000
3	5.025@5.150	4.875	23.350	3.000	2.900	3.000
4	5.025@5.150	4.850	23.350	3.000	2.900	3.000
5	5.025@5.150	4.900	23.400	3.000	2.900	3.000
7	5.025@5.150	4.975	23.650	3.000	2.900	3.000
8	Holiday	5.125	Holiday	Holiday	Holiday	Holiday
9	5.025@5.150	5.250	23.750	3.150	3.000	3.000
10	5.025@5.275	5.200	23.600	3.150	3.000	3.000
11	5.025@5.275	5.300	24.075	3.150	3.000	3.000
12	5.025@5.275	5.350	24.250	3.150	3.000	3.100
14	5.150@5.275	5.375	24.150	3.150	3.000	3.150@3.200
15	5.150@5.275	5.375	24.000	3.150	3.000	3.150@3.200
16	5.150@5.275	5.350	23.800	3.150	3.000	3.150@3.200
17	5.150@5.275	5.275	23.350	3.150	3.000	3.150@3.175
18	5.150@5.275	5.150	23.450	3.000	2.900	3.150@3.175
19	5.150@5.275	5.130	23.400	3.000	2.900	3.150
21	5.150@5.275	5.135	23.350	3.000	2.900	3.150
22	5.150@5.275	5.130	23.200	3.000	2.900	3.150
23	5.150@5.275	5.135	23.150	3.000	2.900	3.150
24	Holiday	5.100	Holiday	Holiday	Holiday	Holiday
25	5.025@5.150	5.075	22.700	3.000	2.900	3.150
26	5.025	5.050	22.750	3.000	2.900	3.150
28	5.025	5.025	22.400	3.000	2.875@2.900	3.150
29	4.975@5.025	5.000	22.125	3.000	2.875@2.900	3.125
30	5.025	5.025	22.500	3.000	2.875@2.900	3.125
Av. for Month	5.131	5.114	23.354	3.050	2.931	3.094
Nov.			**Averages for Week**			
2	5.077	4.900	23.358	3.000	2.894	3.000
9	5.088	4.996	23.500	3.030	2.920	3.000
16	5.181	5.325	23.979	3.150	3.000	3.104
23	5.213	5.159	23.317	3.025	2.917	3.154
30	5.033	5.046	22.495	3.000	2.893	3.140
Nov			**Calendar Week Averages**			
5	5.088	4.892	23.367	3.000	2.896	3.000
12	5.125	5.200	23.865	3.120	2.980	3.020
19	5.213	5.159	23.317	3.025	2.917	3.154
26	5.150	5.104	23.030	3.000	2.900	3.150

Silver, Gold, and Sterling Exchange New York and London

Nov.	Sterling Exchange: "Checks"	Sterling Exchange: "90-Day Demand"	Silver: New York	Silver: London	Gold London
1	3.295000	3.297500	27.125	18.1875	125s 5½d
2	3.312500	3.315000	27.000	18.0625	124s 3d
3	3.291250	3.293750	27.000	18.1250	125s 4d
4	3.291250	3.292500	27.000	18.1875	125s 6½d
5	3.300000	3.301250	27.000	18.2500	.....
7	3.305000	3.307500	27.375	18.2500	124s 9½d
8	Holiday	Holiday	Holiday	18.3750	124s 8d
9	3.303750	3.306250	27.250	18.3750	124s 8½d
10	3.283750	3.286250	27.000	18.1875	125s 4d
11	3.295000	3.296250	27.000	18.1250	125s
12	3.326250	3.328750	27.125	18.1250	.....
14	3.328750	3.332500	27.000	17.8750	123s 2½d
15	3.316250	3.320000	27.000	18.0000	123s 11½d
16	3.316250	3.318750	27.000	18.0000	124s 2½d
17	3.283750	3.286250	26.875	18.1250	124s 10½d
18	3.286250	3.288750	26.750	18.1250	125s 7d
19	3.286250	3.288750	26.750	18.0625	.....
21	3.275000	3.277500	26.750	18.0625	125s 6d
22	3.267500	3.270000	26.625	18.1250	126s 1d
23	3.268750	3.262500	26.625	18.1250	125s 10½d
24	Holiday	Holiday	Holiday	18.1875	126s 10d
25	3.215000	3.216250	26.250	18.1250	127s 8d
26	3.218750	3.221250	25.750	17.9375	.....
28	3.180000	3.182500	25.625	17.8125	128s 4d
29	3.150000	3.152500	25.375	17.8750	129s 3½d
30	3.193750	3.196250	25.500	17.8750	130s 8d
Av. for Month	327.458	.....	26.698	18.099	.....
Nov.		**Averages for Week**			
2	3.28917	.....	26.896	.....	.....
9	3.29825	.....	27.125	.....	.....
16	3.31104	.....	27.021	.....	.....
23	3.27792	.....	26.729	.....	.....
30	3.19150	.....	25.700	.....	.....

Calendar week averages: New York Silver, Nov. 5th, 27.000; 12th, 27.150; 19th, 26.896; 26th, 26.400.

London Market

November	Copper: Standard Spot	Copper: Standard 3 Mo.	Copper: Electrolytic (Bid)	Tin: Spot	Tin: 3 Mo.	Lead: Spot	Lead: 3 Mo.	Zinc: Spot	Zinc: 3 Mo.
1	30.2500	30.4375	35.2500	152.250	152.875	11.8750	12.1250	15.0625	15.3750
2	29.9375	30.1250	34.7500	152.125	152.875	11.6875	12.0000	15.1250	15.3750
3	30.2500	30.3750	34.7500	152.250	153.000	11.7500	12.0625	15.0625	15.3750
4	30.2500	30.4375	34.7500	152.125	152.875	11.8750	12.1875	15.1250	15.3750
7	31.6250	31.8125	35.5000	153.250	153.875	12.1250	12.4375	15.2500	15.6250
8	33.1250	33.3125	36.7500	155.250	156.000	12.4375	12.7500	15.5000	15.8750
9	32.8750	33.1250	37.5000	155.625	156.000	12.5625	12.8750	15.4375	15.7500
10	32.3125	32.5000	37.5000	154.500	155.000	12.4375	12.8125	15.3750	15.8125
11	33.1250	33.3125	37.5000	156.250	157.000	12.6250	13.0000	15.6250	16.0000
14	33.3750	33.6250	38.0000	156.875	157.625	12.6875	13.1250	15.6875	16.0625
15	32.8750	33.1250	37.7500	156.500	157.250	12.4375	12.8750	15.5000	15.8750
16	32.8750	33.1250	37.7500	156.125	157.000	12.4375	12.8125	15.3750	15.6875
17	32.4375	32.6250	37.2500	154.750	155.750	12.2500	12.6250	15.3125	15.5625
18	31.1875	31.3125	36.0000	153.250	154.000	11.9375	12.3125	15.1875	15.4375
21	32.1250	32.3125	36.7500	154.250	155.000	12.0000	12.4375	15.3750	15.5625
22	32.0000	32.2500	36.5000	153.375	154.500	11.9375	12.3125	15.3750	15.5000
23	31.9375	32.1875	36.2500	152.625	154.000	11.7500	12.0625	15.3125	15.5000
24	31.8750	32.2500	36.2500	151.875	153.000	11.6250	12.0000	15.0625	15.2500
25	32.3750	32.7500	36.7500	152.000	153.375	11.6875	12.0000	15.0625	15.1875
28	32.2500	32.6250	37.0000	151.375	152.875	11.6875	11.8750	15.0000	15.1250
29	32.1250	32.5000	37.0000	150.500	151.750	11.6875	11.8750	14.8750	15.1250
30	32.1875	32.5625	37.0000	151.750	153.000	12.0625	12.1250	15.1250	15.3125
Average for month	31.972	.....	36.568	153.585	.....	12.071	12.395	15.264	15.534

The United States quotations are our appraisal of the major markets for domestic consumption based on sales reported by producers and agencies. They are reduced to the basis of cash, New York or St. Louis, as noted. All prices of domestic class are in cents per pound.

Copper, lead and zinc quotations are based on sales for both prompt and future deliveries; tin quotations are for prompt delivery only.

Quotations for zinc are for ordinary Prime Western brands. Zinc in New York is now quoted at 0.35c. per pound above St. Louis, this being the freight differential. Contract prices for High-Grade zinc delivered in the East and Middle West are 1c. above St. Louis

Silver and Sterling Exchange

	New York		London Spot		Sterling Exchange	
	1931	1932	1931	1932	1931	1932
January	29.423	29.780	13.810	19.623	485.260	342.515
February	26.773	30.136	12.432	19.573	485.551	345.141
March	29.192	29.810	13.524	18.336	485.596	363.463
April	28.279	28.298	13.120	16.923	485.764	374.731
May	27.650	27.755	12.858	16.868	486.188	367.370
June	27.250	27.466	12.707	16.844	486.291	364.471
July	28.255	26.700	13.197	16.930	485.349	354.865
August	27.524	27.986	12.815	18.000	485.284	347.449
September	28.180	27.870	14.101	17.998	451.245	347.003
October	29.538	27.195	17.153	17.813	388.029	339.500
November	32.223	26.698	19.393	18.099	371.130	327.458
December	30.120	.....	20.023	.....	336.798	.....
Year	28.700	.....	14.594	.....	452.707	.....

New York quotations, cents per ounce troy, 999 fine. London, pence per ounce, sterling silver, 925 fine. Sterling exchange in cents.

Zinc

	St. Louis		London			
	1931	1932	1931 Spot	1931 3 Mos.	1932 Spot	1932 3 Mos
January	4.035	3.011	12.747	13.113	14.416	14.834
February	4.012	2.817	12.303	12.694	13.872	14.289
March	4.002	2.787	12.190	12.676	12.616	13.024
April	3.717	2.725	11.353	11.838	11.670	11.958
May	3.306	2.532	10.484	10.875	12.432	12.682
June	3.416	2.777	11.270	11.750	11.548	11.866
July	3.893	2.537	12.280	12.802	11.592	11.967
August	3.917	2.758	11.444	12.028	13.594	13.844
September	3.744	3.322	11.571	12.063	15.455	15.616
October	3.377	3.027	12.733	13.216	14.869	15.140
November	3.209	3.094	13.845	14.247	15.264	15.534
December	3.149	.....	14.361	14.818	.....	.....
Year	3.640	.....	12.215	12.677	.....	.....

St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Copper

	F.O.B. Refinery		London Spot			
	Electrolytic		Standard		Electrolytic	
	1931	1932	1931	1932	1931	1932
January	9.838	7.060	44.938	39.459	47.524	46.200
February	9.724	5.965	45.372	36.917	47.950	41.381
March	9.854	5.763	44.818	33.039	47.699	36.786
April	9.392	5.565	42.694	29.943	45.375	34.190
May	8.665	5.237	38.897	28.548	42.175	32.833
June	8.025	5.145	35.827	26.872	38.966	30.841
July	7.698	5.053	34.402	26.071	37.293	29.107
August	7.292	5.219	32.572	31.401	35.388	34.784
September	6.988	5.978	31.503	34.986	36.148	38.318
October	6.775	5.733	34.957	31.890	41.000	36.190
November	6.558	5.131	358.54	31.972	41.190	36.568
December	6.580	.....	38.273	.....	44.409	.....
Year	8.116	.....	38.342	.....	42.093	.....

New York quotations, cents per pound. London, pounds sterling per long ton. Export copper, f.o.b. refinery: Average for November, 5.114

Cadmium and Aluminum

	Cadmium		Aluminum	
	1931	1932	1931	1932
January	67.115	55.000	23.300	23.300
February	55.000	55.000	23.300	23.300
March	55.000	55.000	23.300	23.300
April	55.000	55.000	23.300	23.300
May	55.000	55.000	23.300	23.300
June	55.000	55.000	23.300	23.300
July	55.000	55.000	23.300	23.300
August	55.000	55.000	23.300	23.300
September	55.000	55.000	23.300	23.300
October	55.000	55.000	23.300	23.300
November	55.000	55.000	23.300	23.300
December	55.000	.....	23.300	.....
Year	56.010	.....	23.300	.....

Aluminum in cents per pound, 99 per cent grade. Cadmium, cents per pound.

Lead

	New York		St. Louis		London			
	1931	1932	1931	1932	1931 Spot	1931 3 Mos.	1932 Spot	1932 3 Mos.
January	4.802	3.750	4.604	3.550	13.872	13.905	15.084	15.128
February	4.552	3.712	4.340	3.499	13.444	13.550	14.560	14.571
March	4.527	3.150	4.276	2.993	13.128	13.355	12.345	12.634
April	4.412	3.000	4.164	2.900	12.375	12.606	11.223	11.503
May	3.818	3.000	3.651	2.900	11.491	11.778	10.673	11.036
June	3.917	2.993	3.761	2.896	11.582	11.952	9.608	9.898
July	4.400	2.747	4.225	2.611	12.731	12.899	9.818	10.152
August	4.400	3.235	4.225	3.093	11.944	11.994	11.349	11.588
September	4.400	3.465	4.217	3.315	11.932	12.026	13.122	13.349
October	3.964	3.052	3.773	2.939	13.227	13.270	11.958	12.170
November	3.937	3.050	3.758	2.931	14.577	14.491	12.071	12.395
December	3.792	.....	3.592	.....	15.188	15.361	.....	.....
Year	4.243	.....	4.049	.....	12.958	13.099	.....	.....

New York and St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Antimony, Quicksilver, and Platinum

	Antimony (a) New York		Quicksilver (b) New York		Platinum (c) New York	
	1931	1932	1931	1932	1931	1932
January	7.317	5.976	103.000	64.900	36.000	40.000
February	7.069	6.489	100.205	66.304	34.000	40.000
March	7.127	6.188	100.423	72.537	29.538	40.000
April	6.888	5.746	102.077	72.125	26.346	40.000
May	6.524	5.170	101.140	66.380	24.980	39.500
June	6.342	5.034	92.058	59.481	37.115	37.500
July	6.802	5.000	85.808	53.580	40.000	35.200
August	6.596	5.144	80.115	47.444	40.000	35.000
September	6.542	5.606	76.300	47.500	40.000	33.800
October	6.517	5.575	72.385	47.600	40.000	33.000
November	6.679	5.771	68.587	48.750	40.000	33.000
December	6.231	.....	66.115	.....	40.000	.....
Year	6.720	.....	87.351	.....	35.665	.....

(a) Antimony quotations in cents per pound, for ordinary brands. (b) Quicksilver in dollars per flask of 76 lb. (c) Platinum in dollars per ounce troy.

Tin

	New York		London	
	1931	1932	1931	1932
	Straits		Spot	
January	26.137	21.804	115.798	140.219
February	26.315	22.018	117.919	139.143
March	27.065	21.863	121.852	129.810
April	25.222	19.244	112.775	108.935
May	23.221	20.948	104.331	122.286
June	23.478	19.659	104.966	114.530
July	24.978	20.931	111.478	125.863
August	25.738	22.985	114.875	142.017
September	24.618	24.779	117.813	152.705
October	22.723	23.936	126.932	151.280
November	22.779	23.354	132.857	153.585
December	21.328	.....	138.909	.....
Year	24.467	.....	118.375	.....

New York quotations, cents per pound. London, pounds sterling per long ton.

Pig Iron[1]

	Bessemer		Basic		No. 2 Foundry	
	1931	1932	1931	1932	1931	1932
January	17.50	16.00	17.00	15.00	17.00	15.50
February	17.29	15.68	16.79	14.68	16.79	15.18
March	17.00	15.50	16.50	14.50	16.50	15.00
April	17.00	15.50	16.50	14.50	16.50	15.00
May	17.00	15.50	16.38	14.50	16.44	15.00
June	17.00	14.85	15.50	14.35	16.00	14.75
July	17.00	14.56	15.50	14.00	16.00	14.50
August	17.00	14.50	15.50	14.00	16.00	14.50
September	17.00	14.50	15.50	14.00	16.00	14.50
October	16.92	14.50	15.42	14.00	16.00	14.50
November	16.50	14.50	15.00	14.00	16.00	14.50
December	16.25	.....	15.00	.....	15.75	.....
Year	16.96	.....	15.88	.....	16.25	.....

Iron in dollars per long ton. [1]F.o.b. Mahoning and Shenango Valley furnaces; freight to Pittsburgh, $1.76.

prices for the Prime Western grade.

Quotations for lead reflect prices obtained for common lead, and do not include grades on which a premium is asked.

London prices for lead and zinc are the official prices for the morning session of the London Metal Exchange; prices for copper and tin are the official closing buyers' prices. All are in pounds sterling per long ton (2,240 lb.).

New York silver quotations are as reported by Handy & Harman and are in cents per troy ounce of silver, 999 fine. London silver quotations are in pence per troy ounce of bar silver, 925 fine. Sterling prices represent forenoon market demand. Cables command a premium.

Current Statistics of Production and Stocks of Copper, Lead, and Zinc

Data from American Bureau of Metal Statistics, American Zinc Institute, and Metallgesellschaft.

All Figures Except Tin Represent Tons of 2,000 Lb.

	1931		1932									
COPPER — North and South America	Nov.	Dec.	Jan.	Feb.	March	April	May	June	July	Aug.	Sept.	Oct.
Production, refined, daily average	2,693	2,780	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Domestic shipments	37,436	36,972	.....	.....	.....	.....	.....	.....	.....	(a) 18,300	(a) 19,700	(a) 25,500
Foreign shipments	20,257	26,032	.....	.....	.....	.....	.....	.....	.....	(a) 48,000	(a) 53,700	(a) 55,500
Stocks, blister and refined	.....	722,014	.....	.....	.....	.....	.....	.....	.....	.....	(b) 775,000	(b) 773,000
LEAD — United States												
Production, refined, from domestic ore	31,671	33,576	32,180	28,081	30,345	23,236	25,902	26,068	15,819	17,118	20,498	21,092
Production, secondary and foreign	3,820	4,031	3,554	3,920	3,989	3,134	2,941	2,033	3,754	6,308	2,315	6,984
Production, total, daily rate	1,180	1,213	1,185	1,103	1,107	879	930	936	631	756	760	905
Shipments reported	31,216	30,297	27,867	26,319	31,162	26,081	24,258	21,511	19,723	28,248	26,413	29,764
Stocks, end of month	144,057	151,380	160,257	165,933	169,091	169,091	173,929	180,460	180,255	175,426	171,831	170,171
ZINC — United States												
Production, daily average	684	709	723	742	726	688	601	547	476	432	434	480
Domestic shipments	20,327	21,005	22,472	21,896	22,576	18,046	18,087	14,958	12,896	18,069	21,132	18,801
Export shipments	.....	.....	31	.....	.....	.....	.....	24	.....	39	20	20
Stocks, end of month	130,865	129,825	129,886	129,506	129,451	132,025	132,580	134,032	135,907	131,203	123,056	119,101
World Production Rate (Daily Average)												
Copper	4,070	3,940	.....	.....	.....	.....	.....	.....	.....	.....	(b) 2,500	(b) 2,477
Lead	3,980	3,976	3,938	3,872	3,598	3,587	3,711	3,497	3,190	3,334	3,323	3,529
Zinc	2,725	2,601	2,600	2,615	2,623	2,528	2,392	2,343	2,224	2,121	2,182	2,253
Tin (Metric Tons)	325	351	311	274	340	312	311	261	178	179	253	.....

(a) Deliveries; private estimate. (b) *E.&M.J.* estimate.

Editor: A. W. ALLEN

Associate Editors: A. H. HUBBELL, H. D. KEISER

Assistant Editor: W. N. P. REED

Market Editor: H. H. WANDERS

Publishing Director: H. W. CLARKE

Engineering and Mining Journal

Editorial Correspondents throughout the United States and in
ENGLAND: London
CANADA: Toronto, Sudbury, Victoria
MEXICO: Mexico City
CHILE: Santiago
PERU: Lima
BOLIVIA: La Paz
BRAZIL: Rio de Janeiro
AFRICA: Johannesburg
AUSTRALIA: Melbourne
GERMANY: Berlin
FRANCE: Paris
BELGIUM: Brussels
U. S. S. R.: Moscow

Contents for December, 1932

Copyright 1932, by McGraw-Hill Publishing Company, Inc.
VOLUME 133 NUMBER 12

Editorials		603
Dust Control an Economy Measure	*G. E. Lynch*	606
Limitations of Economic Geology	*C. A. Porter*	609
Summary of Cost of Mining	*M. J. Elsing*	611
Concreting the Denn Shaft	*F. P. Brunel*	614
Developments in Flotation-Cyanidation	*L. E. Tucker and S. J. Swainson*	618
Interpretations of Fault Movements	*D. M. Fraser*	621
Hoist Design and Construction	*Lucien Eaton*	625
Composition of Copper Matte	*C. K. Avetisian*	627
Gravity Milling and Flotation	*J. B. Huttl*	630
Dead-Load Power Consumption	*A. M. Gow and M. Guggenheim*	632
Butane as Fuel	*Rudolf Gahl*	638
Book Reviews		633
Personals		635
Industrial Progress		638
News of the Industry		642
Market Information		648

McGRAW-HILL PUBLISHING COMPANY, INC.
330 West 42d St., New York, N. Y.
Cable Address: McGrawhill, N. Y.

McGRAW-HILL PUBLISHING COMPANY, LTD.
Aldwych House, Aldwych, London, W.C. 2, England
Cable Address: Backlash, Estrand, London

JAMES H. MCGRAW, Chairman of the Board; MALCOLM MUIR, President; JAMES H. MCGRAW, JR., Vice-President and Treasurer; MASON BRITTON, Vice-President; EDGAR KOBAK, Vice-President; H. C. PARMELEE, Vice-President and Editorial Director; HAROLD W. MCGRAW, Vice-President; C. H. THOMPSON, Secretary.

Complete Market News and Prices

THE monthly magazine you are now reading carries only a condensed statistical summary of prices of the major metals. For buyers and sellers of ores, metals, minerals, and scrap who require reliable information as soon as it is available, we now provide *Metal and Mineral Markets*, which goes to press at the close of the metal-market week each Wednesday. It is printed at high speed and is in the mails that evening. In compact form, ready for insertion in a ring binder, it is ideal for desk and reference use. It carries the standard *E.&M.J.* quotations used in contracts the world over, recognized as authoritative and dependable. *Metal and Mineral Markets* is priced to subscribers in the United States and its possessions at $3 per annum; to countries in the Americas but outside the United States, $6; elsewhere, $10 yearly: 52 issues. Address *M.&M.M.*, 330 West 42d St., New York City.

Insist on UPKEEP ECONOMY

. . . fewer delays
. . . quick, easy repairs
. . . permanent service

UPKEEP ECONOMY is difficult to measure in dollars and cents. In your "maintenance costs" an item of only a few dollars may actually represent a loss of several hundred dollars due to delay, lost output, wasted pay roll, and breaking down of smoothly working schedules.

Upkeep economy (best measured in "total maintenance cost" divided by "output") is a vital factor to consider when you select your excavating equipment. To secure upkeep economy you look for balance in strength, simplicity of construction, ample power properly applied, a minimum of moving parts, insurance of permanent alignment, ready accessibility, adequate, convenient lubrication and protection against abrasive dirt.

Bucyrus-Erie gives a maximum of balanced value in all these features. Possibilities of breakdowns are reduced to a minimum. When repairs are necessary they can be quickly made, repair parts are reasonably priced, repair service is prompt and its permanence assured. 572

BUCYRUS-ERIE COMPANY, South Milwaukee, Wis.

Do not purchase any machine — new or used — without careful study on *all* counts. Measure dependability, power, speed, operating efficiency and upkeep economy; weigh the reputation, experience and resources of the manufacturer.

BUCYRUS ERIE

SEE BUCYRUS-ERIE BEFORE YOU BUY!

He found a *fifth* way

PICTURE yourself in this man's position. He knows that four ways of combining flotation and cyanidation have already been proved out and are fast becoming solid text-book history. But he is satisfied that still further economies can be made! And then . . . *he discovers a fifth!*

An important discovery! He introduces flotation at the ball-mill discharge to pull down the gold content of the tube mill circuit at the earliest possible opportunity. By floating out gold at this point . . . before it gets concentrated in the circuit . . . he reduces the values in the classifier overflow by 50% . . . and thus achieves more uniform metallurgy.

This interesting development further emphasizes the fact that somewhere between ball-mill and final tailings disposal, there is a point where flotation may be combined with any one of the older recovery methods to increase yield, cut costs, or both.

And that point can be found . . . in large mill or small . . . by the man who, keeping his mind open and alert, seeks with patience and persistence new ways of applying flotation to the ore which he is milling.

To every metallurgist interested in utilizing the combination of flotation with cyanidation or other recovery methods, American Cyanamid Company offers its help. Cyanamid is familiar with the metallurgy involved, from

AMERICAN CYANAMID

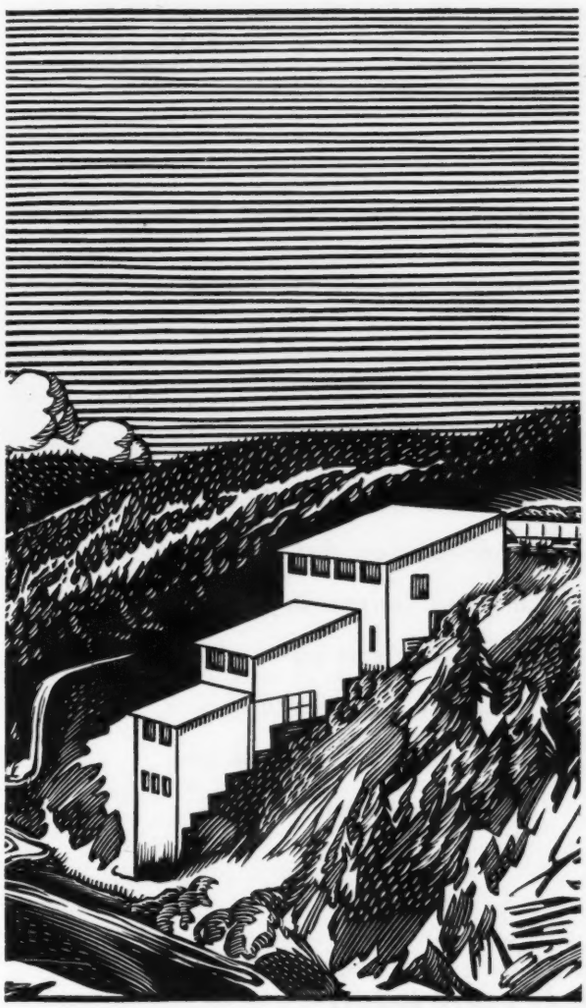
535 FIFTH AVENUE,

AMERICAN CYANAMID SALES COMPANY, INC.,

EUROPEAN DISTRIBUTORS: CYANAMID PRODUCTS LTD., 233 SALISBURY

...to profitably combine and use

Flotation and Cyanidation



Large mines and small, whatever the process of recovery previously used, have demonstrated that the combination of flotation with earlier forms of metallurgy has resulted in higher recovery, lower cost or more uniform tailings. Cyanamid field men stand ready to help you apply flotation to your present recovery process.

laboratory stage to full scale production—has already assisted many operations to increase economies through this new technique. And Cyanamid by means of its complete metallurgical-chemical service makes available to you the needed mining chemicals and the services of its trained field men. Feel free to call on Cyanamid for data on cyanidation-flotation or any other metallurgical problem.

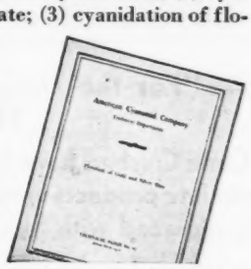
Write for these Technical Papers

To metallurgists and others interested in keeping up-to-date on developments in this interesting technique of milling, we offer two practical technical papers.

"Gold Milling Developments in Northern Ontario" is a down-to-the-minute description of milling practice on outstanding properties in the Kirkland Lake and Porcupine field with hitherto unrevealed details of the metallurgy used and interesting sidelights in unique application of flotation in several different ways.

Technical Paper No. 20, "Recent Developments in Flotation and Cyanidation of Precious Metal Ores", details the actual practice of a number of properties which have combined flotation and cyanidation.

It covers (1) flotation as an adjunct to cyanidation; (2) cyanidation of the flotation concentrate; (3) cyanidation of flotation tailings; (4) cyanidation of gravity concentrates; (5) amalgamation and flotation; (6) amalgamation and cyanidation; (7) sand leaching; etc., etc.



We will gladly send you a copy of either or both papers. The coupon is for your convenience.

COMPANY

NEW YORK

AZUSA, CALIFORNIA

HOUSE, LONDON, E. C. 2, ENGLAND

American Cyanamid Company
535 Fifth Avenue, New York City

Send me a copy of:

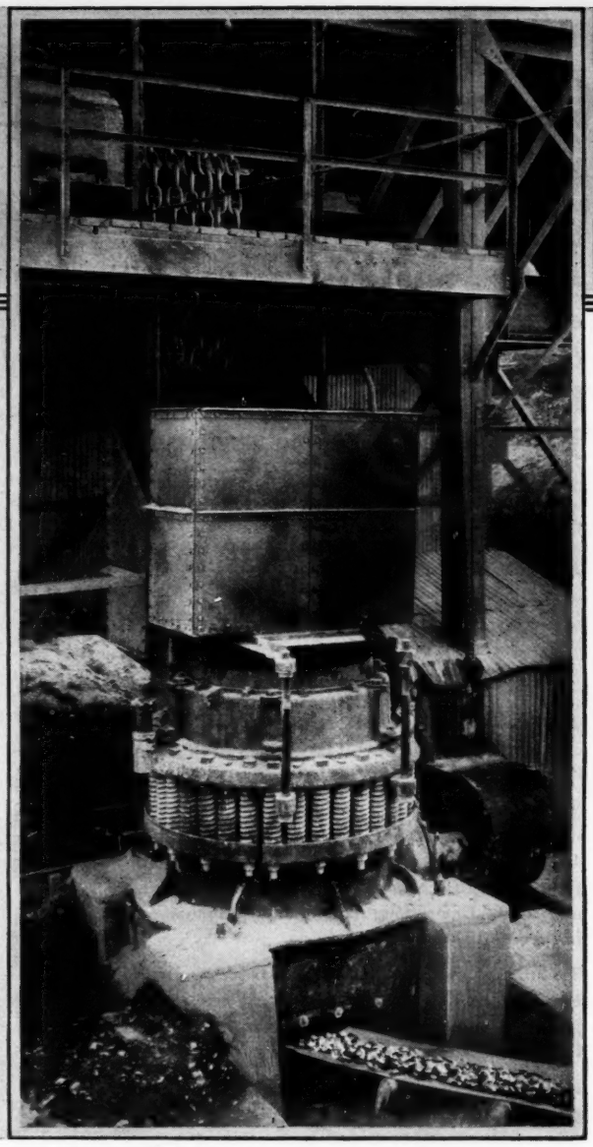
☐ "Gold Milling Developments in Northern Ontario"

☐ Technical Paper No 20, "Recent Developments in Flotation and Cyanidation of Precious Metal Ores."

Name_____

Address_____

SYMONS CONE CRUSHERS



—ANSWER THE REDUCTION CRUSHING PROBLEM WITH— PERFORMANCE

This 4 foot installation is typical of the hundreds of Symons Cone Crushers that are doing the reduction crushing at mines, stone, slag and gravel plants. There are five sizes to choose from, one that will meet your capacity requirement.

"For the reduction crushing, a Symons Cone has been installed." This is a statement commonly heard today.

Symons Cone Crushers have been installed in most of the newer modern crushing plants built in recent years that deliver fine products in quantity. About 90 percent of the obsolete crushers taken out of existing plants are being replaced with Symons Cones.

There is a reason for the rapid trend toward the Symons Crusher and that reason is PERFORMANCE. Its capacity for fine crushing is unequalled. Its ability to deliver a big tonnage at low cost makes it use imperative to show profitable operation in these days of close competition and more rigid material specifications.

NORDBERG MFG. CO. MILWAUKEE, WISCONSIN

LONDON, WC2, ENG., Bush House — NEW YORK, 51 E. 42nd St. — LOS ANGELES, 1462 Stanley Ave.

AUSTRALIA
Noyes Bros., Sydney,
Noyes Bros. Pty. Ltd., Melbourne

ENGLAND and INDIA
Fraser & Chalmers Eng. Works, Ltd.
Erith, Kent, Calcutta

NEW ZEALAND
John Chambers & Son, Auckland,

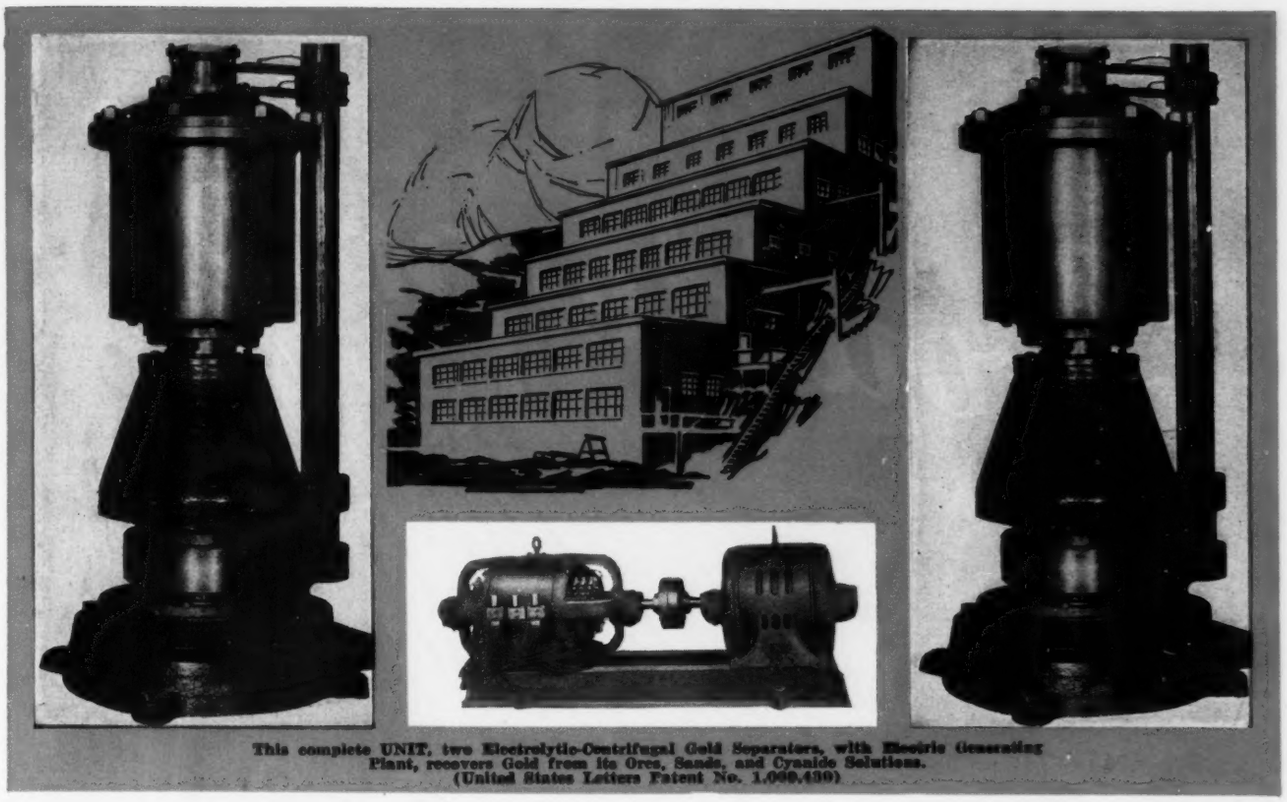
DENMARK
Haehre & Co.,
Oslo, Norway.

SWEDEN, NORWAY and FINLAND
Morgardshammars Mek Verkstads,
Morgardshammar, Sweden.

SOUTH AFRICA
Fraser & Chalmers (S. A.), Ltd.,
Johannesburg

ITALY
Loro & Parisini
Milan, Genoa and Rome

SPAIN
Gumersindo Garcia, Madrid



This complete UNIT, two Electrolytic-Centrifugal Gold Separators, with Electric Generating Plant, recovers Gold from its Ores, Sands, and Cyanide Solutions.
(United States Letters Patent No. 1,040,430)

Thank you, E. & M. J. readers, for your enthusiastic reception ▲ ▲

Can you recover *your gold* more profitably, and in higher percentages, through the Thayer Process?

In response to our September announcement in this Journal we have received inquiries from all parts of the world and have upwards of 100 negotiations for installations now under way. A complete installation of Thayer equipment is now being built for a prominent California gold mining company which has been looking for just this improvement in gold processing technique.

The Thayer Electrolytic-Centrifugal Gold Separator recovers the smallest particles of Gold and Platinum from their sands and ores. It also recovers Gold from its solutions in cyanide, etc.

In any normally-equipped mill, the only additional equipment required, to profitably recover the Gold, are two (2) Thayer Electrolytic-Centrifugal Gold Separators, with the Special Complete Electrical Installation, required for generating and controlling the electric current for recovering the Gold.

As the Separator revolves, a floating wall of mercury is formed on the interior surface of the outer cylinder, which constitutes the Cathode for Electrolytic Precipitation. The insulated inner cylinder is the Anode.

The pulp or solution passes, under pressure, continuously through the annular space between the two cylinders, in direct contact with the activated mercury, and, at the same time, the electric current passes through the pulp or solution, in its course from Anode to Cathode. Occasionally the mercury is drawn from the machines and retorted, thereby recovering the Gold.

By the powerful combined and simultaneous forces of Electrolytic Precipitation and Centrifugal Force, the particles of Gold, Platinum, Iridium, etc., are irresistibly drawn, precipitated, and absorbed into the Cathode wall of mercury, the Gold instantly amalgamating with the activated mercury, and the platinum metals (if present in the pulp) are held in the mercury by its electrification, the waste material passing through to the dump.

The Thayer Process is practically continuous:—There is no loss of mercury, and the extraction efficiency is higher than by any other known process.

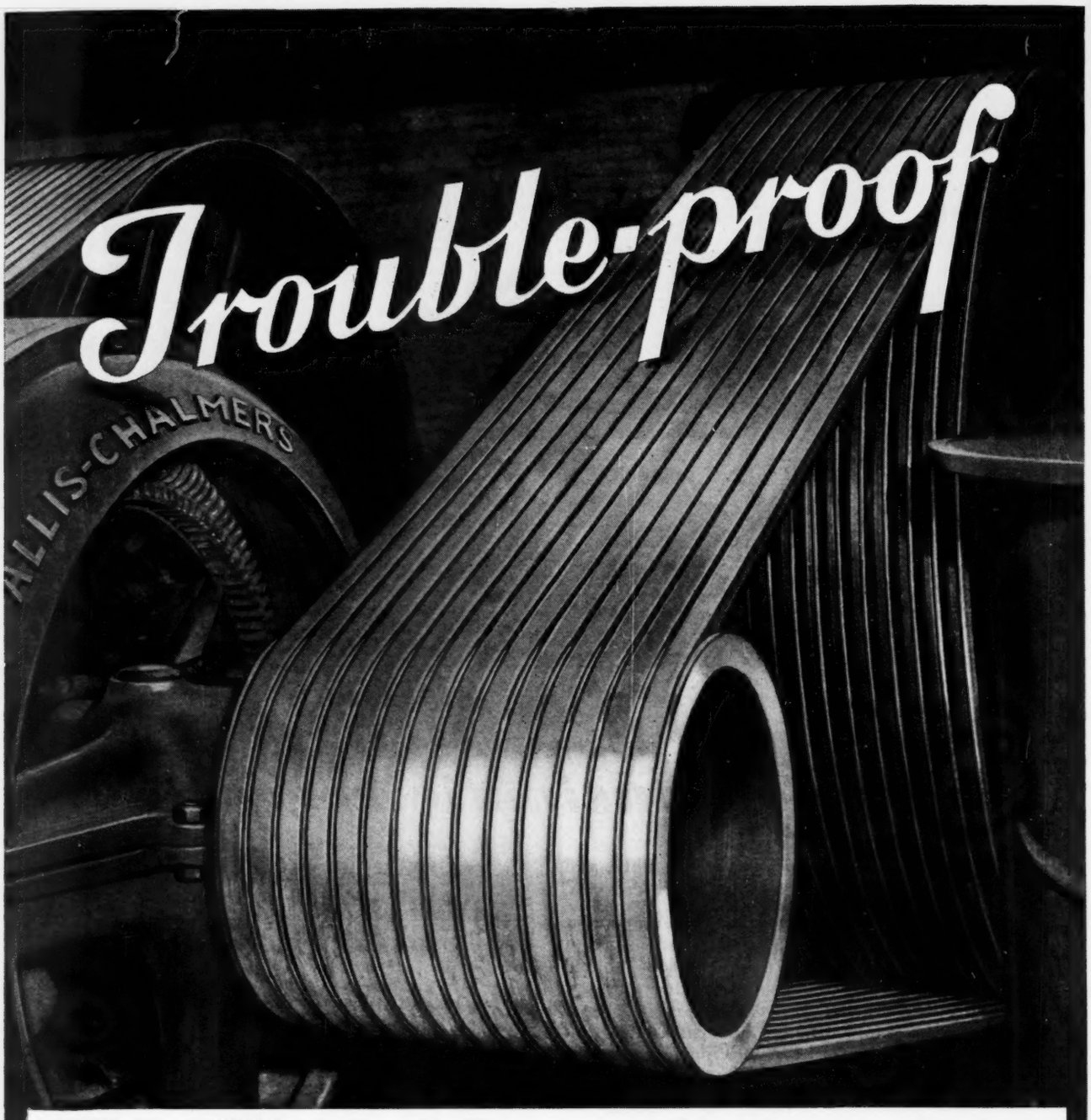
Complete information furnished promptly on request.

RUSSELL THAYER

M.Am.Soc.C.E.

Consulting Mining Engineer

230 ALLEN LANE, PHILADELPHIA, PA., U. S. A.



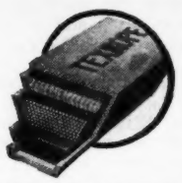
Texrope Multiple V-Belt **Drives** assure continuous operation because they are designed and built to absorb shock loads; to operate in the presence of water or moisture; to throw off foreign matter coming in contact with the belts; to grip tighter when the load is increased. Texrope Drives do still more. They eliminate oiling, greasing, belt dressing, bearing failures, and other attendant costly maintenance. Texrope Drives have demonstrated repeatedly that they not only prevent trouble but institute a substantial saving in floor space and in power bills year after year.

For Texrope Bulletin 1228-K write the nearest district office or Allis-Chalmers Manufacturing Company, Milwaukee, Wisconsin.

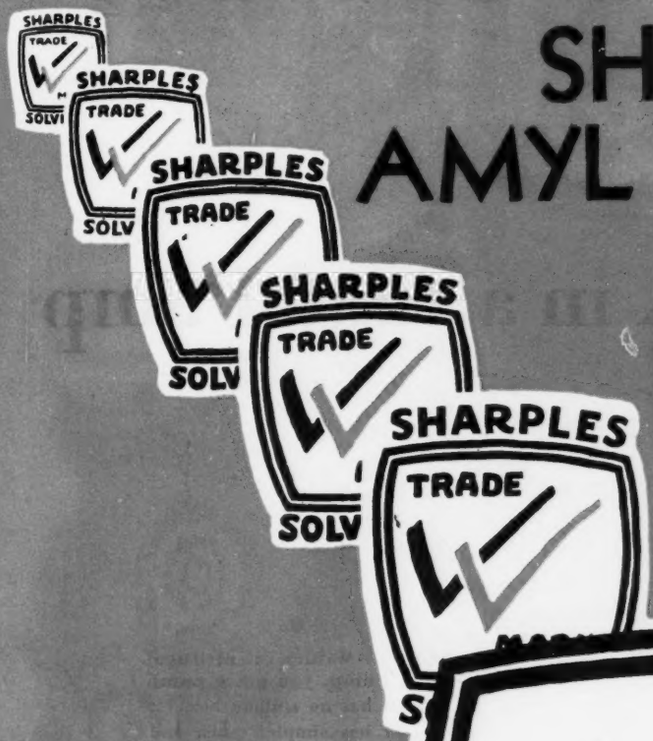
ORIGINATED BY ALLIS-CHALMERS

TEXROPE DRIVES

THE DRIVES THAT REVOLUTIONIZED TRANSMISSION PRACTICE • • •



Texrope Belts are made by B. F. Goodrich for Allis-Chalmers. Complete stocks are carried in Allis-Chalmers warehouses all over the globe. When your belts need replacement the nearest Allis-Chalmers Office will serve you promptly.



SHARPLES AMYL ALCOHOLS FOR FLOTATION REAGENTS

Are You Interested In Higher Recoveries?

PENTASOL AMYL XANTHATE has "proven it's place" as an extraordinary reagent for preventing high tailing losses of fine gold in gold flotation operations.

On new and difficult flotation problems PENTASOL AMYL XANTHATE should be carefully tried. It is so powerful in its flocculating and collecting action that substantial gains in both recovery and grade have been made by its use.

Samples of PENTASOL or of the commercial AMYL XANTHATES, made from PENTASOL, will be gladly sent on request.

DOUBLE-CHECKED FOR PURITY AND UNIFORMITY

THE SHARPLES SOLVENTS CORP.

2324 Westmoreland Street — Philadelphia, Pennsylvania

JUST THINK!

No Stuffing Box in a Sand Pump

RIGHT there is a story of low maintenance costs. It emphasizes the reduced attention and servicing that need be given the pump.

And it is this feature of the Wilfley Pump above all others that influenced the leading mining enterprises the world over to buy it, try it out and then standardize on it as the best means for handling sands and slimes.

This "no stuffing box" feature of the Wilfley Pump alone merits your choice. Other features serve to emphasize its low pumping costs.

●

In the Wilfley Centrifugal Sand Pump, you get a pump

1) that has no stuffing box
2) that has simple, quick and positive clearance take up
3) that makes use of extra heavy pumping parts
4) that is so designed that pumping parts can be changed in a few minutes.

●

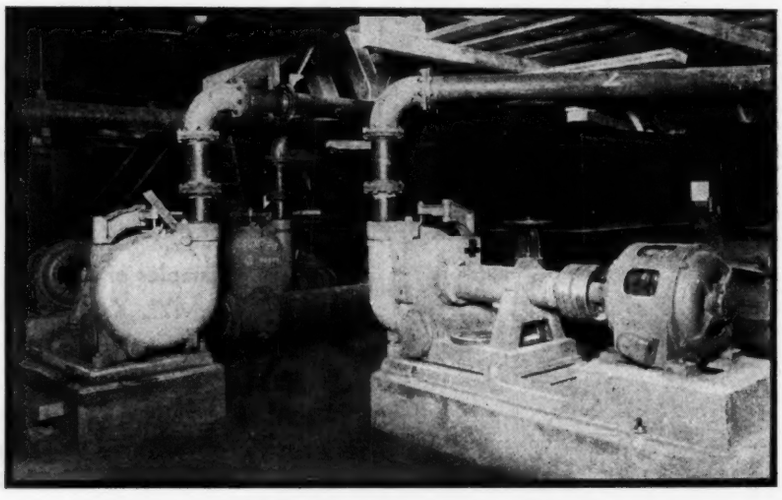
A catalogue gives complete details.

A. R. WILFLEY and SONS
INCORPORATED
Denver, Colorado, U. S. A.

Canada—
The Mine & Smelter Supply Co.,
c/o Wm. Hamilton, Ltd.
Licensees and Manufacturers
Peterborough, Ont.

Australia—
The Mine & Smelter Supply Co.,
c/o F. H. Jackson,
67 York St., Sydney,
Licensees and Manufacturers

South Africa—
Edw. L. Bateman (Pty), Ltd.,
Locarno House, Johannesburg,
Licensee and Manufacturer



WILFLEY

centrifugal SAND PUMPS

PATENTED

P E R F O R M A N C E

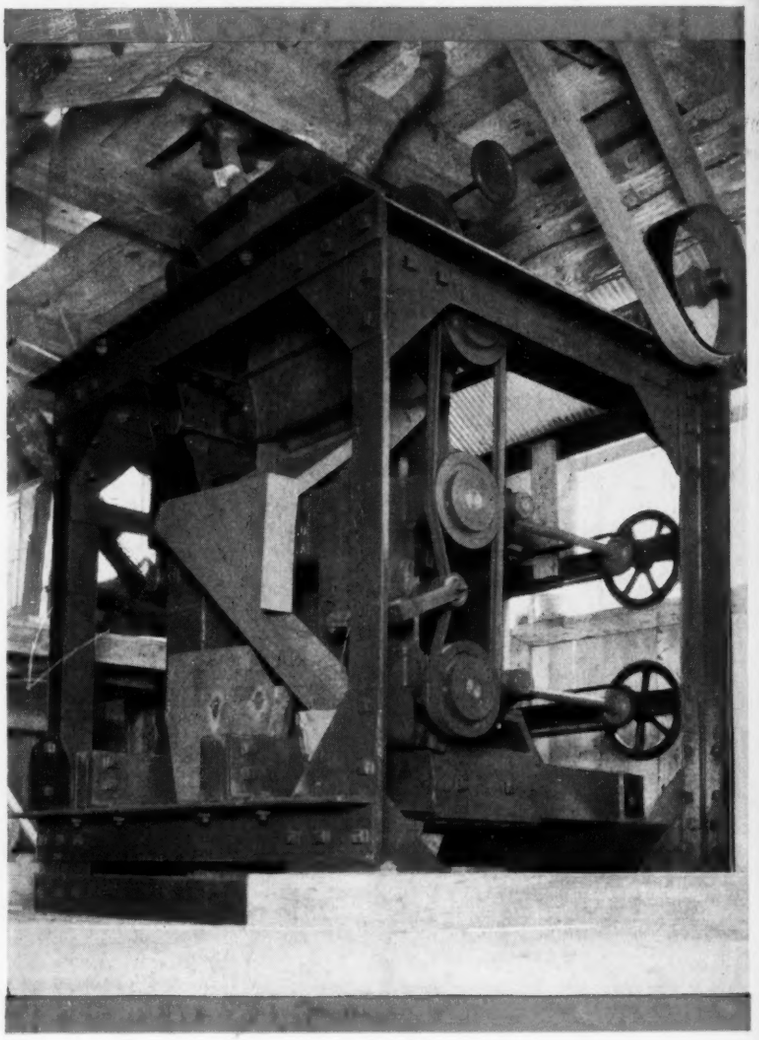
GREAT WESTERN ELECTRO-CHEMICAL CO.

9 Main Street, San Francisco Plant at Pittsburg, Calif. Cable Address: "Chlor"
New York: 225 Broadway Rhodesia, Africa: Johnson & Fletcher, Ltd.
El Paso, Texas: H. J. Baron Co. Montreal, Quebec: Wm. J. Michaud Co., Ltd. Vancouver, B. C.: Robert S. Carter

UNIV OF MICHIGAN
GENERAL LIBRARY
ANN ARBOR MICH

STEARNS SIGNIFIES
SUPERIORITY IN
MAGNETIC EQUIPMENT

TITANIUM
Chooses
STEARNS TYPE "K"
as most efficient

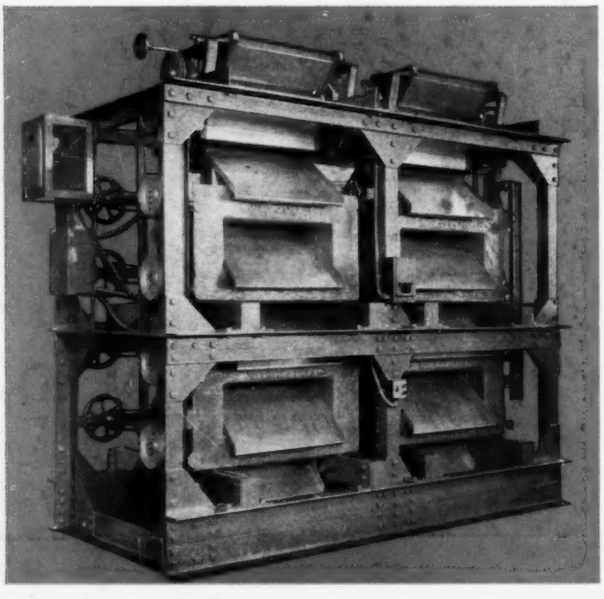


Titanium Corporation of America wanted an efficient magnetic separator for titanium concentration. The unit was needed at the mill for working a new titanium ore deposit at MALVERN, Arkansas.

Titanium is a difficult ore to concentrate magnetically, and it was only after careful investigation of the whole field of magnetic separators that Titanium Corporation selected the Stearns Type K Induction Separator as most efficient for the service required. The unit installed, as shown in the illustration, is a three pole Stearns Type K Induction Separator, built complete and delivered in fourteen days by the Magnetic Mfg. Company.

The remarkable efficiency of the Stearns Type K is due to its ability to provide an intense magnetic field with low power input and correspondingly low power consumption.

On any job of magnetic separation, Magnetic engineers are well qualified to serve you with thirty years' experience designing and building magnetic equipment: separators, conveyor magnets, drums, chutes, clutches and special equipment. Use our complete laboratory facilities for magnetic separation tests and possible application.



MAGNETIC MFG. COMPANY
624 S. 28th St.
MILWAUKEE, WISCONSIN

HIGH STEARNS DUTY
MAGNETIC SEPARATORS

One of the largest induction separators ever built, 10 pole Stearns Type K weighing 30,000 lbs. Used for processing and concentrating glass sand.